Practical Short Cuts in Milling and Metallurgy

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Time and Labor Saving Devices That Have Proved Their Worth

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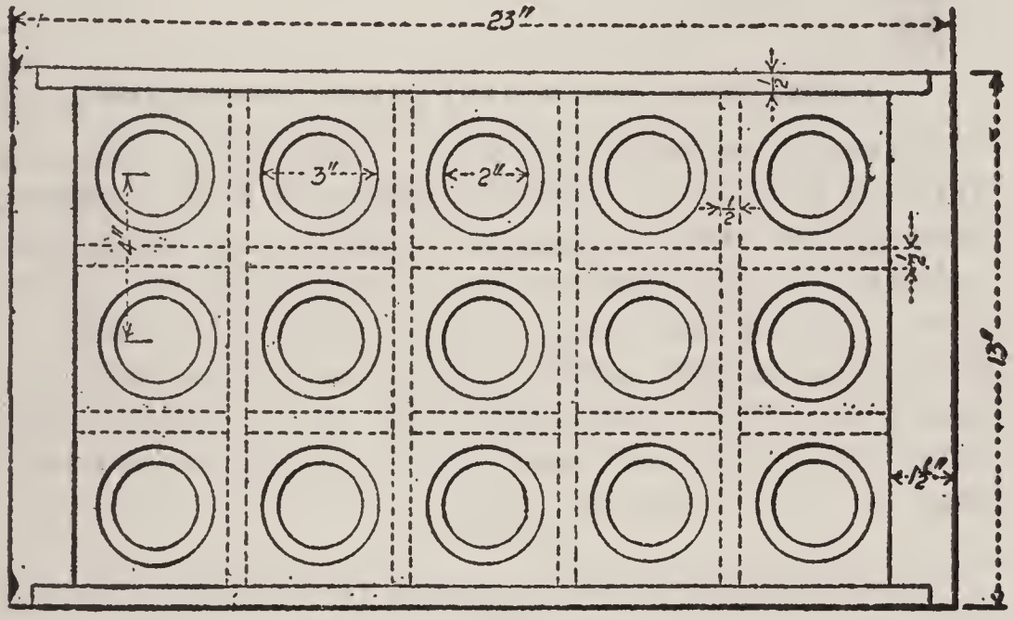
Racks for "Slop Copper" Flasks

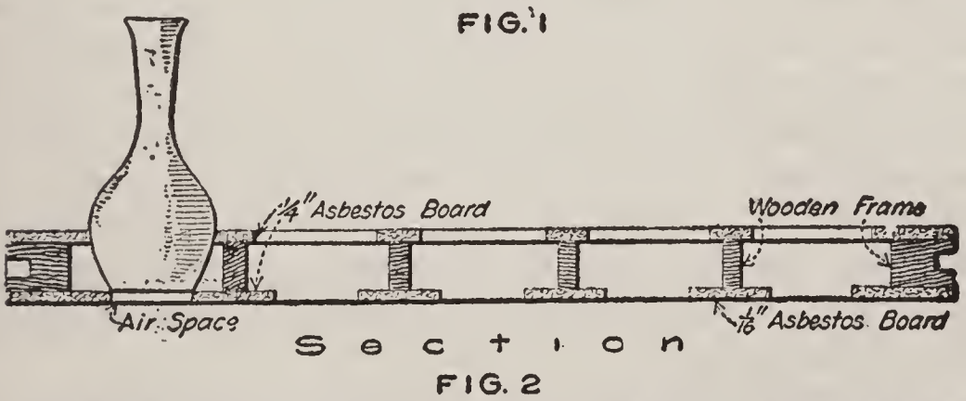
A handy "kink" in the manipulation of the cyanide method was developed in the Miami copper laboratory some time ago by F. C. O'Brien. It consists of sets of decomposing trays and drying racks for the flasks by the use of which handling of individual flasks is almost entirely eliminated and much time is saved. The decomposing tray (Figs. 1 and 2) consists of a wooden frame covered both top and bottom with $\frac{1}{4}$-in. asbestos board. Circular holes are cut in the top to allow the flasks to pass through the asbestos but still to fit snugly. In the bottom strip of asbestos, smaller holes are cut on the edges of which the flasks rest, having air space between them and the hot plate. If the temperature of the hot plate is high, the entire bottom of the tray is covered with $\frac{1}{16}$-in. asbestos as shown.

The drying rack is made entirely of wood with holes corresponding exactly to those in the decomposing tray.



Plan

FIG. 1



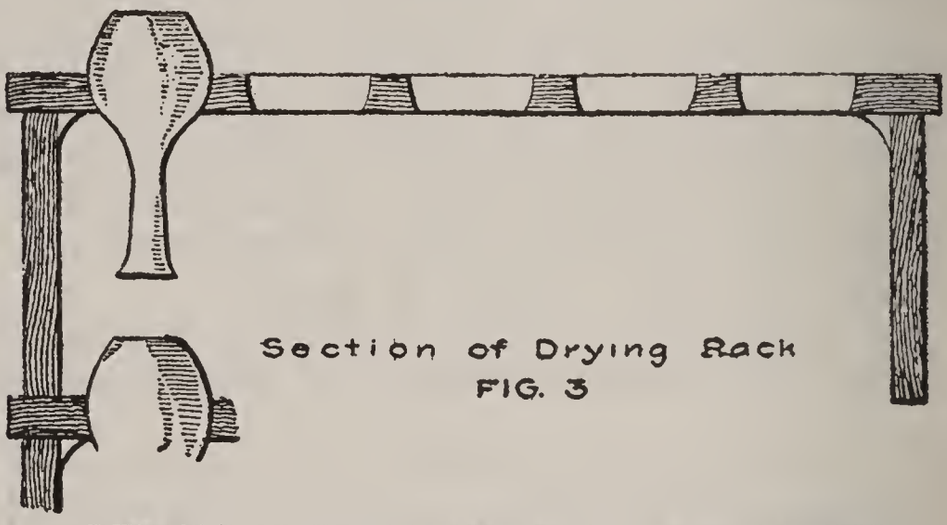
Section

FIG. 2

DECOMPOSING TRAY USED IN MIAMI LABORATORY

The supports are long enough to allow several racks to be stacked one upon the other. A convenient place to dry and store the flasks and racks is in a closet under the hot plate. This can be fitted with guides and the racks placed on these guides. The flasks are then out of sight and out of the way when not in use.

Manipulations with racks and trays are few and simple. After washing, the flasks are inverted in the racks and allowed to drain. When needed, a tray is placed over the flasks in the rack so that the bottoms of the flasks enter the holes in the tray. The rack and tray are then inverted together and the rack lifted off leaving the flasks upright in the tray. The samples are then weighed into the flasks, acid added and the tray placed upon the hot plate. After decomposition, water and



DETAILS OF TRAYS AND RACKS FOR FLASKS

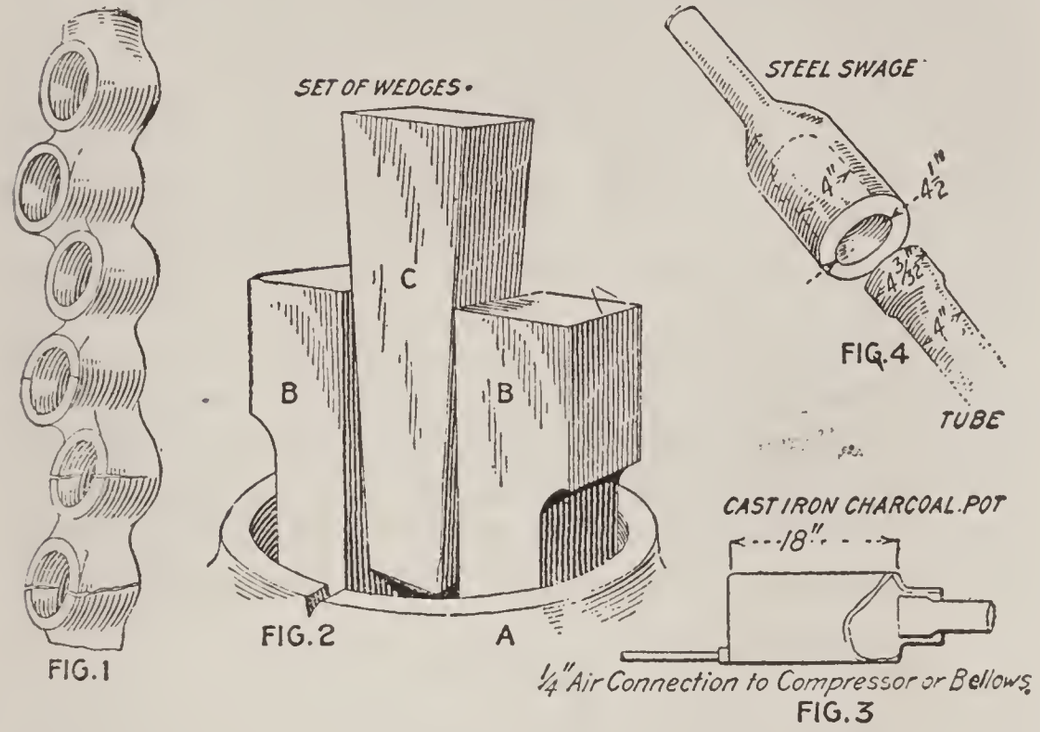
ammonia are added to the flasks while still in the tray. The only handling of individual flasks is in transferring them to the burrette rack, in titrating and in washing and placing the flasks in the drying racks. In this laboratory the trays are made to fit the 14 x 24-in. hot plates. Each tray holds 15 flasks and all samples are run in sets of 15. A standard ore is run in the middle flask (No. 8) of each and the factor so determined is used for the entire set.

Renewing Cast-Iron Headers in B & W Boilers

We are constantly being reminded of the high cost of fuel and the rapidly increasing cost of supplies, labor and the like. It is not only necessary for the engineer of a power plant to economize along the lines mentioned above, but he must also cut the cost of repairs to a minimum. Any plan that will tend to do away with costly repairs, therefore, should meet with particular favor at this time, and the following is written in the belief that it will prove of practical utility at many mine power plants.

In a certain boiler plant consisting of several batteries of Babcock & Wilcox boilers, considerable trouble was experienced from the breaking of the cast-iron headers. The boilermaker's bill reached such proportions that it became necessary to devise some means of renewing headers without outside help. The following method is now used in performing this work, and it has proved quite satisfactory:

Fig. 1 shows a nine-tube, vertical, cast-iron header. In case such a header cracks, or is broken, the first thing to do is to free the damaged header from the



FIGS. 1 TO 4. METHOD AND TOOLS FOR BREAKING OFF THE HEADER AND SWAGING DOWN THE TUBES

tubes. Fig. 2 shows a set of wedges used for this purpose. The face of the handhole is nicked deeply, the side wedges put in place and then the wedge *C* driven in until the header breaks. This method usually, but not always, makes a clean, square break, leaving the tube end free. After all the tubes are clear, the iron around the circulating tube and mud-drum nipple can be broken with a hammer.

Fig. 3 is a cast-iron charcoal burner, 18 in. long and 8 in. in diameter, with a connection for air to a bellows or an air compressor. After a good fire is started in the burner, the end is placed over the end of a tube. When the end of the tube is heated to a cherry red, the swage, Fig. 4, is placed over the end and sledged until the tube end is brought to its regular size. The diameter of the hole at the end of the swage is large enough so that the enlarged end of the tube will enter, but is reduced a short distance into the regular size of the tube.

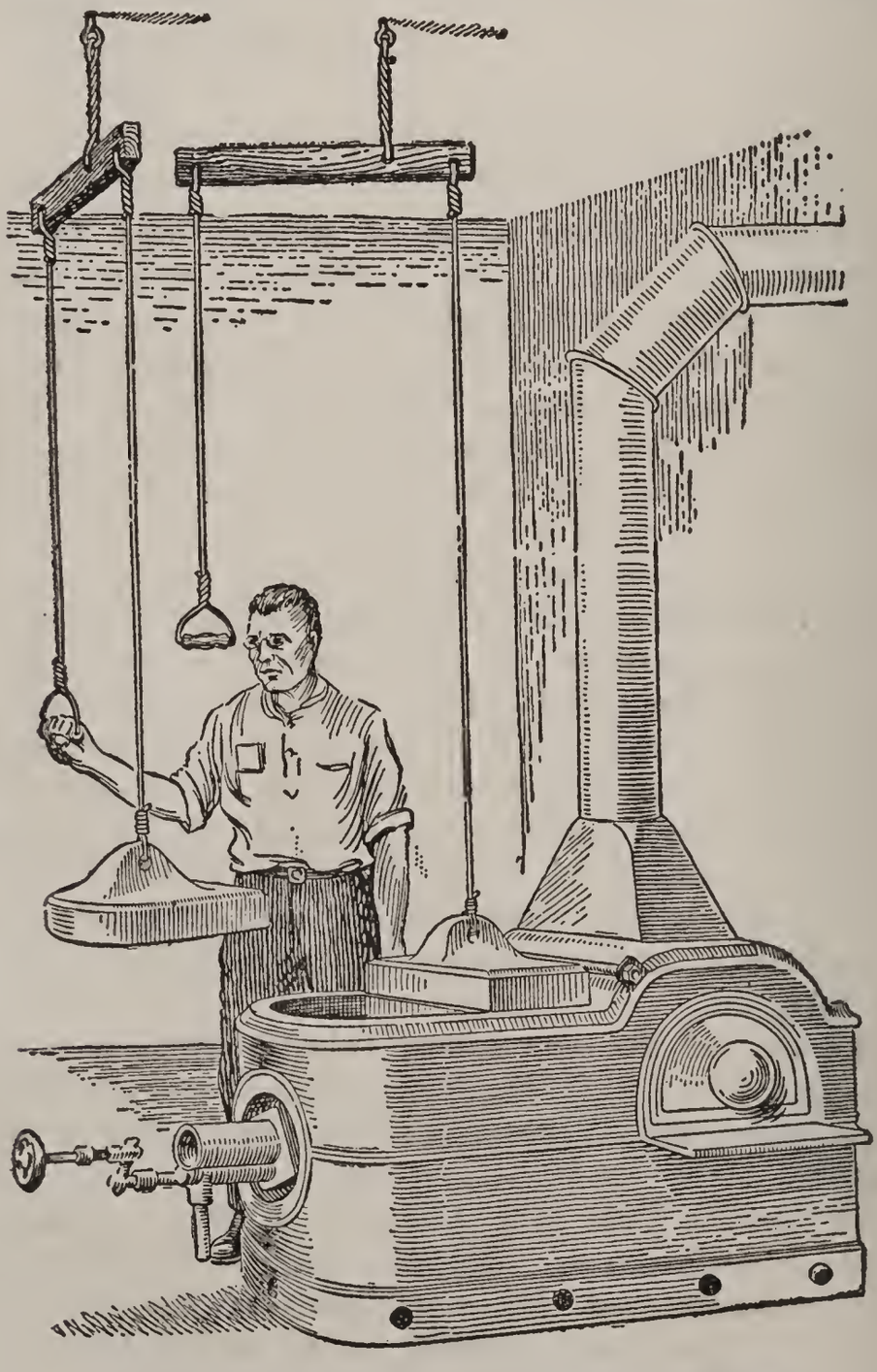
The end of the circulating tube is brought to size the same way. It saves time to use a new mud-drum nipple since the old nipple is usually damaged when removed.

For heating the nine tubes of a header about 30 quarts of charcoal is required. This is much cheaper than replacing new tubes and having the old tubes re-ended. Also, with a repair outfit of this kind much time is saved.

Device for Lifting Furnace Covers

In the usual combination assay furnace the lids or covers of the crucible section supplied by the manufacturers are often frail and of short life. When they burn through, they can be profitably replaced by large slabs of soapstone or fireclay, which can usually be obtained at the nearest hardware store. These are durable and conserve the heat. The difficulty with them is that they are so heavy that they are not easily handled. I solved this difficulty in my own practice by rigging up a lever.

This consists simply of a piece of board 1 x 4 in. by 2 ft., bored with three holes as shown in the accompanying sketch. Baling or other wire is used for suspending it from the ceiling and for attaching it to the furnace cover. The holes should be so placed in the board that the center one will be 1 in. from the upper edge and

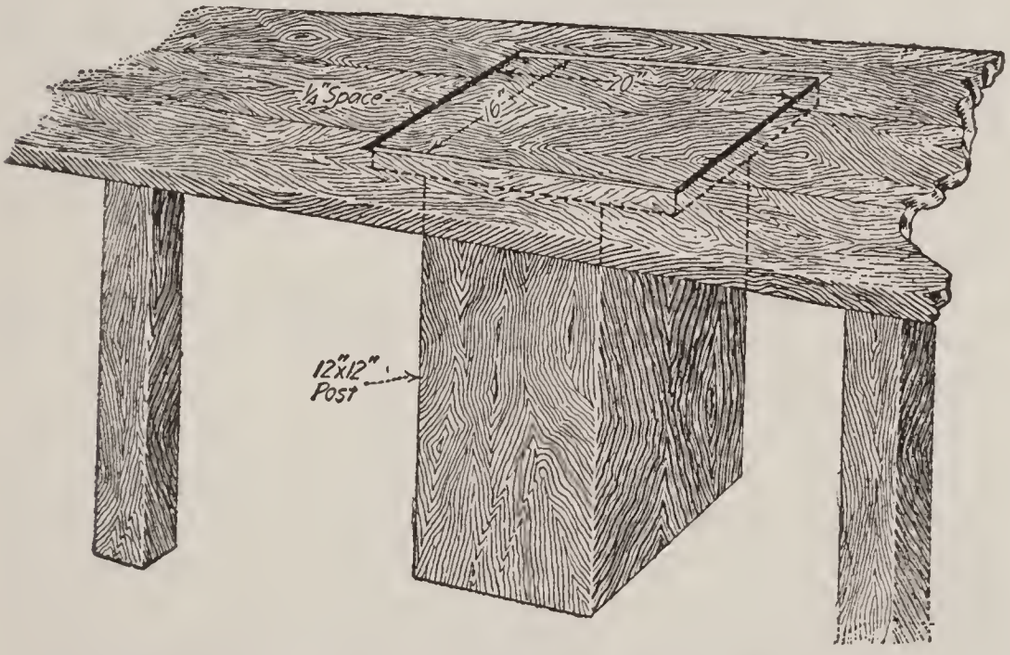


DEVICE FOR LIFTING FURNACE COVERS

the other two 1 in. from the lower edge. The longitudinal position of the central hole will depend on the weight of the cover; it must be sufficiently near the wire attached to the latter to make a gentle pull all that is necessary to lift the cover. The board should be hung a little to one side so that the slab will swing gently away when raised and rest on one edge of the furnace. With the handle in one hand, the tongs in the other will easily guide the slab in any direction. The wire may be attached to the cover by means of a bail, which acts like a pair of ice tongs. It is made of $\frac{1}{4}$-in. iron, the ends pointed to fit in holes cut in the edge of the slab above the center, so as to give better poise. They should be chipped out carefully just deep enough to hold; do not try to drill, but chip them out carefully with a nail and light hammer.

Table for Assay Balance

Care is always taken in an assay office, or should be, to place the balance on a table or support so constructed as to be free from vibration. The special construction usually consists of heavy posts or columns supporting the table top and passing through the floor without



BALANCE TABLE SECURE FROM VIBRATION

touching it. This prevents vibration of the table due to any jarring of the floor, but unless the table is one of massive construction, as, for example, a slate slab on brick pillars, vibration or movement of the balance will be caused by touching or leaning on the table. To prevent absolutely all movement of the balance stand, and yet secure cheapness in construction, the table shown in the illustration was built. It consists simply of a small table top, 20 x 16 in., supported on a 12 x 12-

in. post passing down through the floor with the lower end buried deeply in the ground. The hole in the floor is slightly larger than the post. Around this small table is built a larger one which is supported independently. Its size may be varied to suit conditions or requirements, and that of the small table top may be made to conform to the size of balance used. This method of supporting is easily adapted to any number of balances in the weighing room and is more effective than the sign "Please do not lean on this table."

Changing a Krupp Dry-Grinding Mill Into a Wet-Grinding Mill

The mill of the old Calumet & Massey mine near Massey, Ont., was taken over by the Kenyon Copper Mines, Ltd. Previously, water concentration and the Elmore process had been in use. The mill was rearranged to suit the Callow flotation process. It was found, after several runs, that the capacity could be increased except for a Krupp mill that acted as a throttle on the whole process. While it was not expedient to buy any new machinery at the time, it was desirable to have greater capacity.

The ore from the mine contained a certain amount of moisture and, when ground fine in the Krupp mill, formed a cement and choked the screen, cutting down the capacity of the machine. There was also some ore stocked on the surface and in the wintertime, when a little snow became mixed with it, the Krupp mill would choke altogether.

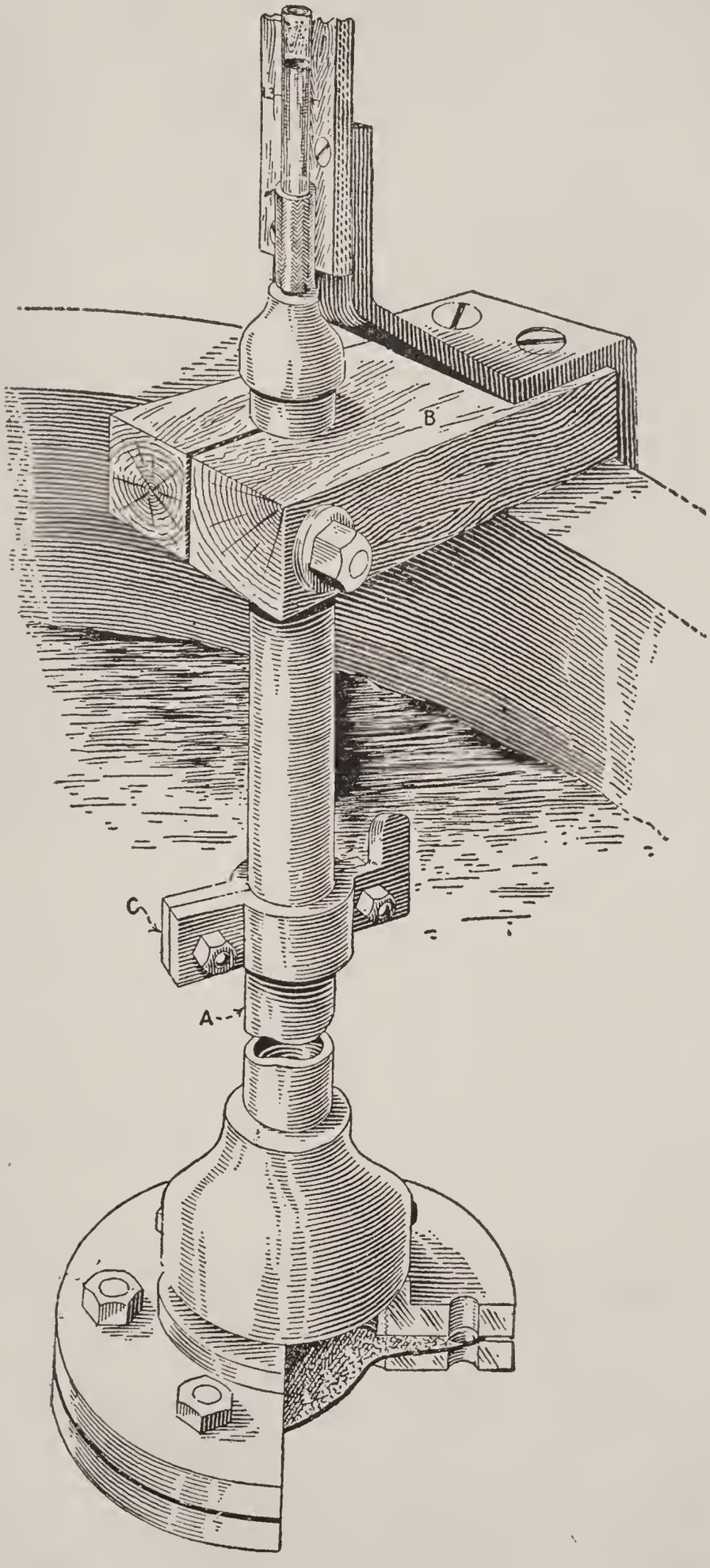
It was suggested that this Krupp mill be run wet. Therefore, during a shutdown, the change was made from the dry to the wet mill.

The casing was taken off the Krupp mill altogether. A large iron tank was placed under the drum and blocked up as high as possible. A lead was then taken from one end of the tank. An iron pipe was used for this purpose and was run into a launder. At the top of the drum was placed the water supply, being simply a water pipe with a tee and a header arranged with three feeds on each side of the tee.

Wet-grinding proved to have other advantages besides increased capacity. From the top of the Krupp mill there was a stack to take care of the dust. The dust loss was considerable, and the wet arrangement, of course, stopped it entirely; also further damage to men and machinery from dust in the mill air.

A Simple Pulp Indicator

An appliance designed to show at all times the specific gravity of pulp under agitation in a Pachuca tank has been tried out with successful results. It is shown in the accompanying illustration. *A* is a 1-in.



PULP-DILUTION INDICATOR

iron pipe of such length that its lower end can be submerged 10 ft. in the pulp while the upper end is about 6 in. above the surface. The pipe is suspended from the wooden clamp *B*, which rests on the top of the tank. At the lower end of the pipe a 2-in. to 1-in. reducer is attached, and to this, by means of a nipple, a flange is screwed, across the opening of which a piece of cotton filter cloth is stretched. Another flange is then bolted on to hold the cloth. A clamp *C* is fixed in such a position that its horn, as shown in the sketch, is exactly 10 ft. above the surface of the filter.

The air lift of the tank is shut off for a few seconds, and when the surface movement has subsided, the pipe is adjusted in the clamp *B* until the top of the horn on *C* is just on a level with the pulp surface. The pipe at the top is now reduced to $\frac{1}{4}$ in. by means of a plug and nipple. With a short piece of rubber tubing the nipple is connected to a length of glass tubing supported vertically on a board, the lower end of which rests on *B*.

The zero point is on the pipe level with the horn on *C*. At a point, say, 2 ft. above *C* there is marked on the board a line designated at 1.2; a point 1 ft. above that will be 1.3, and so on. The distances betwen these points is marked off in intervals of $\frac{1}{10}$ ft. If required, these latter may be further subdivided.

Now, assuming that the pulp in the Pachuca tank has a specific gravity of 1.45, the clear solution in the glass tube will be forced up to a point 4.5 ft. above the zero mark; or in other words, a 10-ft. column of pulp at 1.45 specific gravity will exactly support a 14.5-ft. column of water or solution at a gravity of 1.

It is essential that all joints in the apparatus be perfectly tight, for should slimes leak in at any part of the pipe near the bottom, the specific gravity of the column in the pipe would rise above 1 and a reading lower than the true figure would be recorded. Should there be a leak at the top of the pipe at the junction, say, of the iron pipe with the glass tube, a low reading would be recorded, for the filter might not be able to pass solution quickly enough to maintain equilibrium.

Formation of Quicklime in Roasting Ores from Manhattan, Nev.

About three years ago, while experimenting on the base antimonial gold ores from Manhattan, Nev., it was noticed on roasting the ore that there was formed a large amount of quicklime which not only interfered with the extraction secured by cyaniding, but also separated from the solutions, coating the laboratory appa-

ratus which came in contact with the same. Recently some experiments were made to determine the effect of different roasting temperatures on the amount of quicklime produced. The ore used in the tests contained the following minerals: Quartz, calcite, pyrite, stibnite, limonite and traces of realgar, orpiment, cervantite, kermesite and basic iron sulphates and carbonate. An analysis of the ore showed approximately 18 per cent of calcite. The roasting tests were carried out on ore ground to —80-mesh and were conducted in a muffle with free access of air.

The results of the tests are shown in condensed form in the table. The first test was started in a cold muffle

QUICKLIME FORMED IN ROASTING MANHATTAN ORE

Test	Maximum Temperature C°	Length of Test in Minutes	Free Lime in Roasted Ore.%	Free Lime per Ton, Lb.
1.	650	52	0.8	16
2.	Below 600	52	0.2	4
3.	720	52	2.4	48
4.	760	40	3.2	64
	830	90	4.9	98
5.	790	40	4.0	80
	845	90	5.3	106

and brought up to a maximum temperature of 650 deg. C. in 20 min. and held approximately at this temperature for 52 min. Sample analyzed 0.8 per cent free lime or 16 lb. free lime per ton of roasted ore. Test No. 2 was conducted in the front of the muffle when test No. 1 was being run, but was heated just enough to ignite the ore and roasted with little help from the muffle. Length of test, 52 min. Sample analyzed 0.2 per cent free lime, or 4 lb. per ton of roasted ore. Test No. 3 was conducted at a maximum temperature of 720 deg. C., reached in 20 min. and continued for 52 min. Sample analyzed 2.4 per cent free lime or 48 lb. of free lime per ton of roasted ore. Test No. 4 was conducted at temperatures ranging from 630 deg. C. to 830 deg. C., total length of test 90 min. A first sample was taken at the end of 40 min., when the maximum temperature was 760 deg. C., showed 3.2 per cent free lime, or 64 lb. per ton of roasted ore. The final sample showed 4.9 per cent free lime, or 98 lb. per ton of roasted ore. Test No. 5 was conducted at muffle temperatures ranging from 680 deg. C. to 845 deg. C., total time 90 min. A first sample, taken at the end of 40 min., maximum temperature 790 deg. C., showed 4.0 per cent of free lime, or 80 lb. per ton of roasted ore. The final sample showed 5.3 per cent of free lime, or 106 lb. per ton of roasted ore.

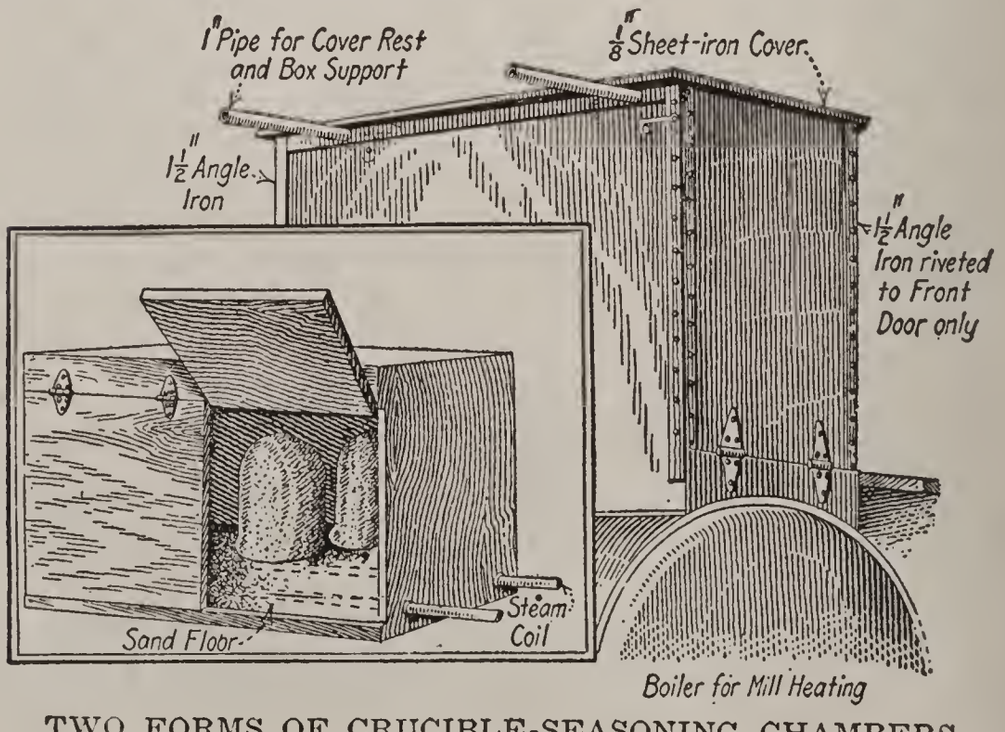
As calcium carbonate begins to decompose below a red heat it would probably be impossible to give an ordinary roast to such an ore without forming some

quicklime; while at the higher temperatures excessively large amounts of quicklime are formed. It would be expected that this excessive amount of lime would cause trouble in several ways. The usual amount of lime in treating gold ores running from one to five pounds per ton of ore treated, such amounts as indicated by the above tests, if present, would probably cause the formation of scums and possibly foaming, filling up of pipes, coating of filters, coating of zinc during precipitation, and retard the solution and precipitation of the gold. From a large number of tests made in the laboratory it appears that to treat this ore successfully it must be roasted, and that during roasting the temperature must not be carried to too high a degree or prolonged beyond the proper length of time or excessively large amounts of quicklime will be formed, interfering seriously with the treatment of the ore.

Crucible-Seasoning Chests

Considering the high cost and comparatively short life of the domestic crucibles used in refining gold and silver precipitates, it is advantageous to take all precautions in lengthening their life. We have inside and outside corrosion, cracking and other troubles. The least expensive and probably greatest life extender for crucibles is proper seasoning, especially when they have become damp in transit or storage.

At the Commonwealth Mines, Pearce, Ariz., a wooden chest, with steam coils covered with sand, is used. The chest holds four crucibles, and by the time required, they are thoroughly seasoned. Since using this chest the life of a crucible has been increased from 10 to 25 per cent, which, considering the initial cost (between $50 and $60), amounts to quite a saving.



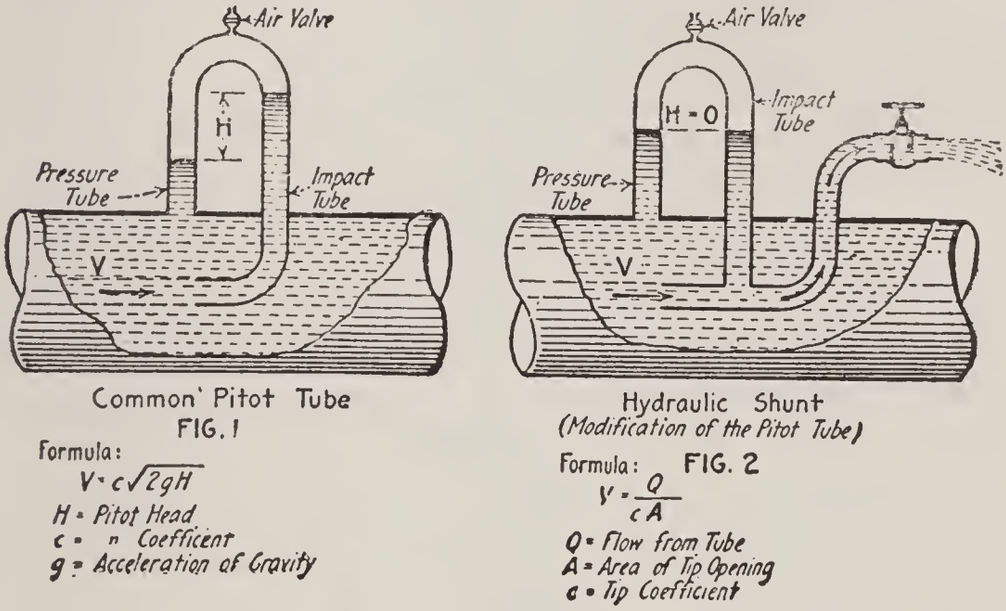
TWO FORMS OF CRUCIBLE-SEASONING CHAMBERS

At the Churchill Milling Co.'s mill, Wonder, Nev., a sheet-iron seasoning box is placed above the mill heating boiler. As shown by the sketch, the top and front sides open on hinges, and the crucibles are lifted out by means of chain blocks.

The Nevada Hills Mining Co. at Fairview, Nev., keeps two or three crucibles in the bullion vault and inside of each is placed a 110-volt electric-heating element. This latter method is successful, though it takes a considerably longer time to season and costs more.

Modified Pitot Tube

The pitot tube has long been used as a device for measuring the flow of liquids and gases, but only when used with the utmost care have the results proved uniform. Many experimenters have worked with modified forms in the endeavor to reduce the variation in



ALMOST AS SIMPLE AS A PITOT TUBE

results, but it is evident not only that the data obtained are variable, but that the same tube may have different coefficients.

In order to correct this latter defect, Prof. H. A. Thomas has devised the "Hydraulic Shunt-Flow Tube." This is a tube so arranged that it may be introduced into the stream with the tip directed against the flow and yet maintain at the tip the same pressure that existed before the introduction of the tube. The water flows into this tube and may be shunted into a small container and weighed, leaving the velocity undisturbed.

The velocity of flow at the tip of the tube will be equal to the quantity of water collected in the measuring tank, in a measured time, divided by the top area. It is possible to demonstrate mathematically that turbulent flow should not affect the coefficient of the tip. The tip coefficient should be unity under all conditions, but experiments show that it varies less than 1 per cent.

Volume of Material in Conical Piles

Inasmuch as the angle of repose of ore in piles averages about 37 deg., it is a simple matter to develop a formula that will give the volume in cubic feet by performing a simple multiplication. The formula is

$$Y = 0.0995d^3$$

where Y is the total number of cubic feet in the pile and d is the diameter of the base of the pile in feet.

To make the operation still simpler the writer has

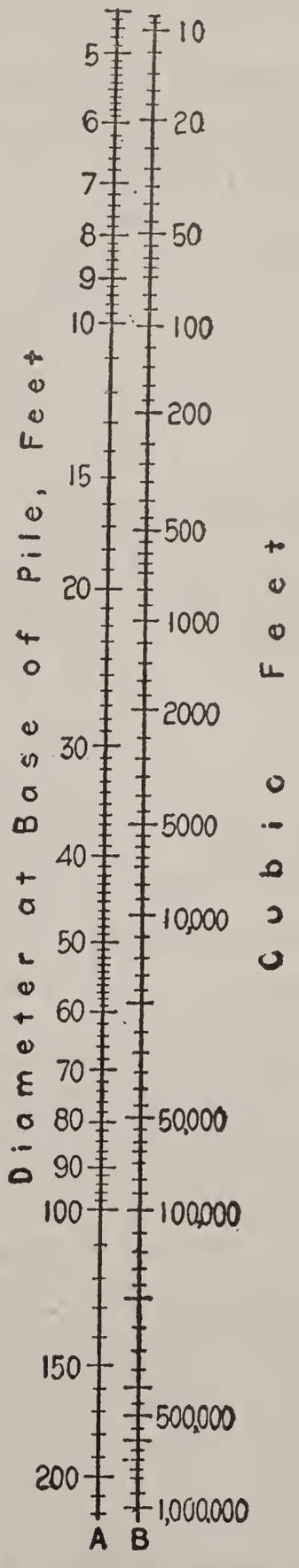


CHART FOR DETERMINING VOLUME OF MATERIAL IN CONICAL PILE

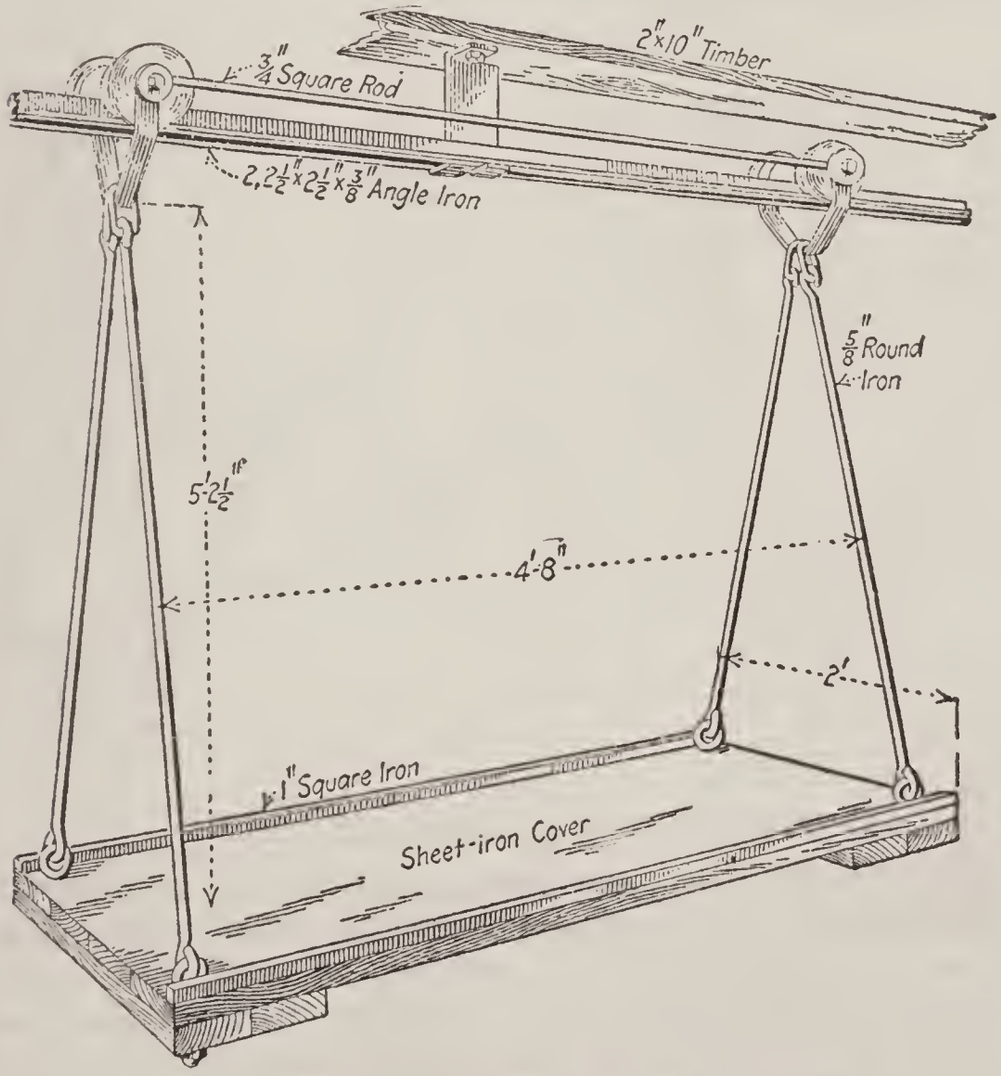
developed the accompanying chart, in the use of which one simply has to glance across from column *A* to column *B*, or vice versa.

For example: How many cubic feet of iron ore in a conical pile whose base diameter is 30 ft., The answer is: About 2700 cu.ft. Inversely, if it is desired to know the extent of the plot of ground on which, say, 10,000 cu.ft. may be piled, one has simply to glance in the other direction—from column *B* to column *A*—and there's the answer: About 46.5 ft. in diameter at the base.

The range of this chart, it will be observed, is wide enough to cover piles that are most ordinarily found in practice, running from 5 to 200 ft. in diameter at the base.

Battery Carriage for Supplies

The carriage for battery supplies shown in the accompanying illustration was built in the shops of the Aurora Consolidated Mines Co., Aurora, Nev., and

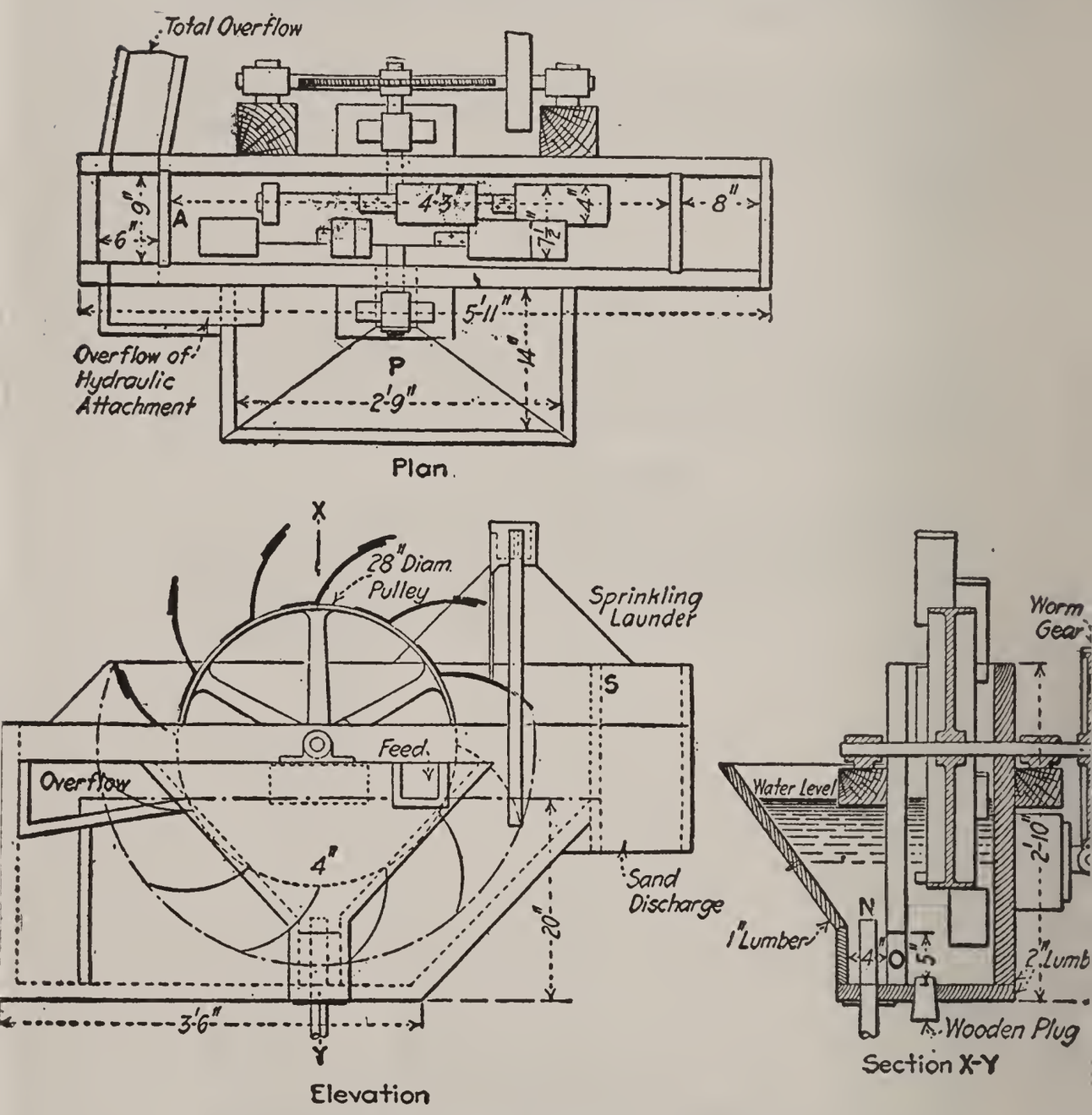


CARRIAGE FOR BATTERY SUPPLIES

has proved to be quite a success in handling the heavy dies, shoes, bossheads and false dies to and from the forty 1,550-lb. stamps. It was designed by Mr. Coutoure, former master mechanic of the company.

Combination Pulp Classifier

The requirements of the milling practice at a tin-concentration plant were such that neither the existing hydraulic nor mechanical classifiers completely satisfied, so a new machine was designed to do the work. As shown in the accompanying drawing, the device consists of a narrow wooden trough in which a paddle-wheel revolves. Details of the construction of the classifier can be easily gathered from the drawing. The



COMBINATION PULP CLASSIFIER

tips of the blades are protected by ¼-in. iron strips fixed with two countersunk rivets. The strips last about 4 months and are replaced at small cost. The blades are cut from ¼-in. plate and bent in the fire. They are fixed to the rim of the pulley by three 1 x ⅜-in. machine bolts. The holes in the rim and those in the blades are bored from a templet, and spare blades are kept ready so that but little time is lost in making renewals. The wheel is driven at the rate of 3 r.p.m. by a simple worm gear—in this case taken from an old

vanner—directly belted to a linshaft. The feed is introduced into the trough near the bottom at *O*, after having been submitted to hydraulic classification in the pyramidal attachment *P*, in which an ascending current of water flows out of the pipe *N*. Consequently the material undesirable in the jig feed is floated off with the use of less hydraulic water than needed in an ordinary hindered-settling classifier because the slimes that come with the sands into the trough are separated by the paddlewheel and flow over at *A*. It is evident that classification in the hydraulic pocket does not entirely take place under hindered-settling conditions, because in that case the sand treated by the wheel could not contain any slimes. The amount of hydraulic water used is such that the dilution of the overflow does not exceed the required density and therefore the classification cannot be perfect. However, the most objectionable factor—namely, the presence of slimes in the jig feed—is to a large degree eliminated by the subsequent action of the paddlewheel. The moisture in the sands discharged by the wheel can be regulated by raising the level of discharge or by increasing the distance between the tips of the blades and the discharge.

The discharge level of the sands can be varied by inserting strips of wood of different heights into the slots marked *S*. To obtain practical results the sands are discharged at a point 1½ in. above the overflow with 50 to 55 per cent moisture, but with the level of sands discharge raised to 2½ in. above the overflow and the discharge edge spaced from the wheel, as shown in the drawing, the product contains but 30 to 35 per cent moisture, although under these conditions the capacity decreases out of all proportion. If the machine is overfed, the sands pile up so high at the discharge that part of them remain lying on the paddles and fall back into the trough at the other side, finally choking the classifier.

The second method gives a drier and cleaner product, because the sands are piled up in front of the wheel until they reach a sufficient height to slide over the discharge edge, and nearly all the water drains back into the trough. There is no objection to placing the overflow rim in the trough close to the paddles, which makes the machine more compact.

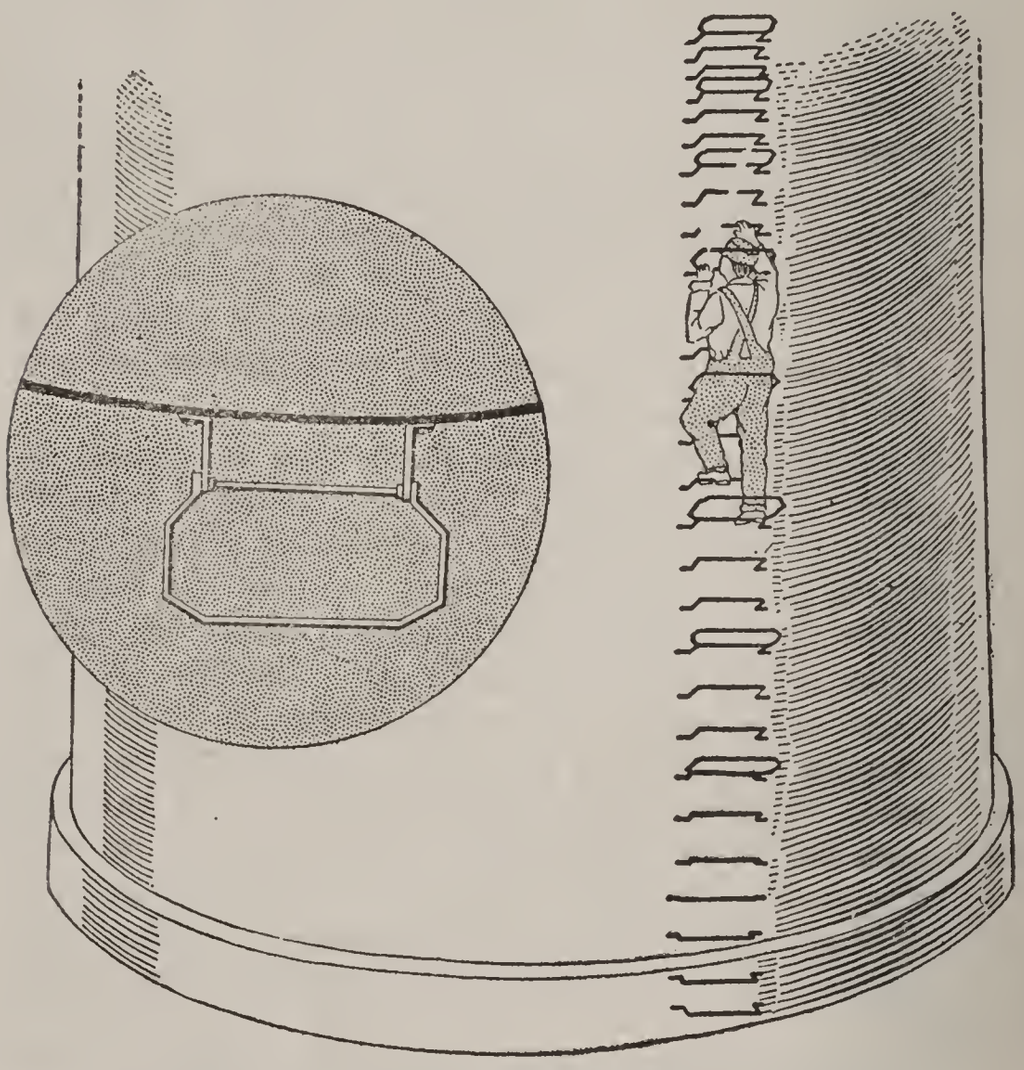
A spray is provided to remove the adhering grains of sand from the blades of the wheel, and after having cleaned the blades, the water falls on the pile of sands lying in front of the wheel and gives them a final wash. The slower the wheel revolves the cleaner will be the sands product, because then every blade brings up slightly more sands than it can deliver, and part of the

material falls back into the trough to be shoved up again by the next blade. Therefore the sand is repeatedly turned over before being discharged, which materially assists in removing the slimes.

The classifier can easily handle, per 24 hr., 60 tons of dry pulp with a specific gravity of 3.5 to 3.8. It satisfactorily performed the work required of it.

Safety Ladder for Smoke-Stacks

Ladders built on the outside of smoke-stacks have little attraction for the fellow who has to climb them occasionally. The man who has climbed an ordinary



LADDER PROVIDED WITH SAFETY GUARDS

ladder to the point where he keenly realizes that his "dear life" depends entirely on his muscles, and has then climbed on to the point where he realizes that his muscles are failing and quivering from sheer exhaustion appreciates the value of a ladder inclosed in guards. It is almost impossible to fall from this ladder, as the enveloping hoops or guards are not more than three feet apart, so at no time is a man's body entirely devoid of protection. The hoops are large enough to give plenty of clearance when climbing, yet small enough so that one can lean back against them in perfect safety while resting hands and arms.

Ore Sampling at a Custom Mill

Buying ore for a custom mill in a superannauted district is not an enviable avocation. This is especially true where decades of imposition on the part of ore buyers have developed a suspicious public. The Argo mill is one of several operating exclusively on custom ore. To allay the suspicion so deeply noted in the

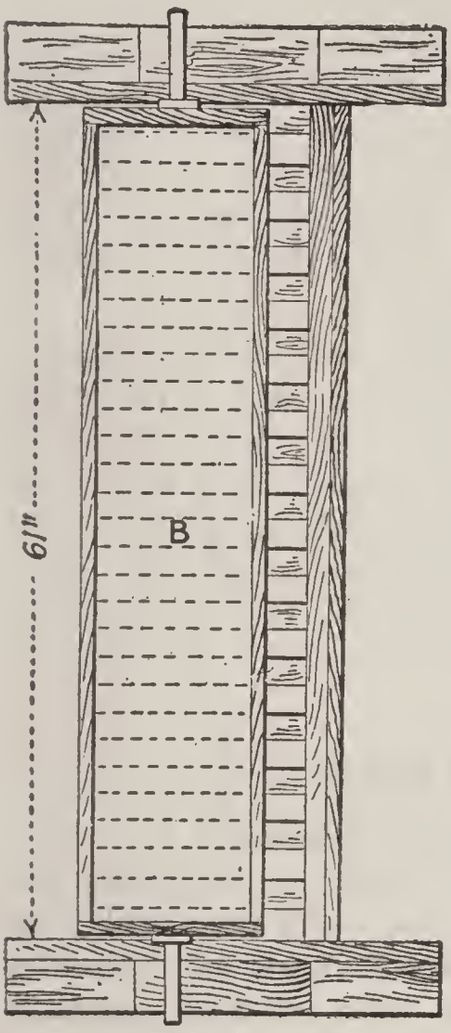


FIG. 1

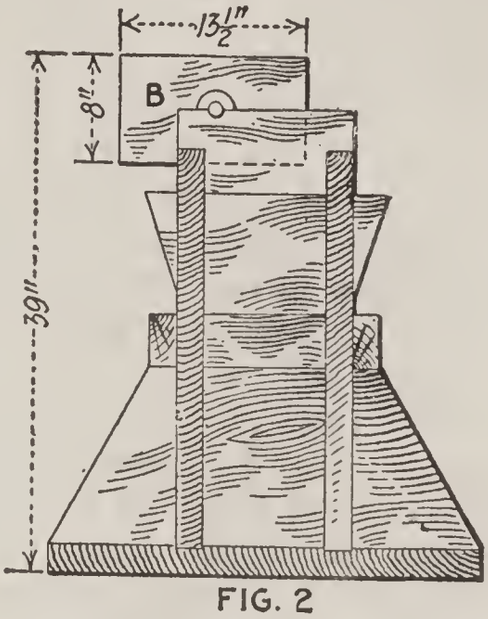


FIG. 2

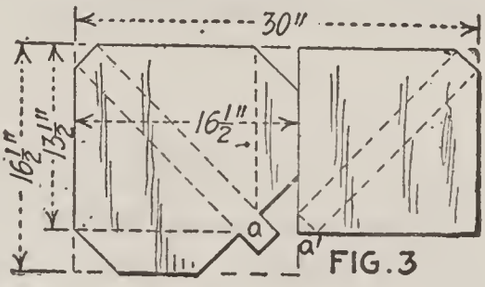


FIG. 3

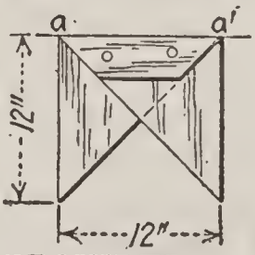


FIG. 4

SAMPLING ARRANGEMENTS PROVIDED

district, we have endeavored to maintain a sampler free from the possibility of manipulating to our advantage.

The prevalent Western custom of sampling ore is to crush and pile the entire lot and shovel it over, setting aside every tenth shovel for coning and quartering. The possibility of error, intentional or otherwise, in this system is evident to one familiar with it. Instead of removing a tenth by shovel, we remove a fourth from a continuous stream of ore by means of a Snyder sampler. This operation is twice repeated automatically and continuously, the ore being crushed finer in each operation. A final sample of from 500 to 4000 lb. results. It has been customary with us to reduce this sample by the cone-and-quarter method. This method is open to serious error, as has been pointed out by

the Bureau of Mines, and further, it is possible for an expert sampler to juggle the sampling, as has been pointed out by most of the shippers. To eliminate this objectionable feature, we have installed a cutter of the Jones, or Brunton, type. There is nothing particularly new in it except that it is larger than usual. This makes a sampler flow sheet as nearly free from manipulation as possible. It has gained the confidence of the shipper to an extent greater than anticipated. Whereas, formerly about 10 per cent of the lots were resampled, only one has been resampled in the last 250 lots.

Fig. 1 is a plan of the cutter and Fig. 2 an end section. A box *B*, with a capacity of 200 to 250 lb., is filled, leveled and emptied upon the cutter. The operation is repeated until the sample is small enough to be handled on smaller cutters.

The sampler was made by the men at the plant. No. 18 iron was found suitable for the cutting arrangement. Fig. 3 shows the method of cutting a piece of 30-in. sheet iron. If cut along the solid lines and bent along the broken lines, the two resulting pieces may be bolted or riveted together, as shown in Fig. 4. Projections, *a* and *a'*, fasten it rigidly to the framework.

Spiral Tube-Mill Feeders

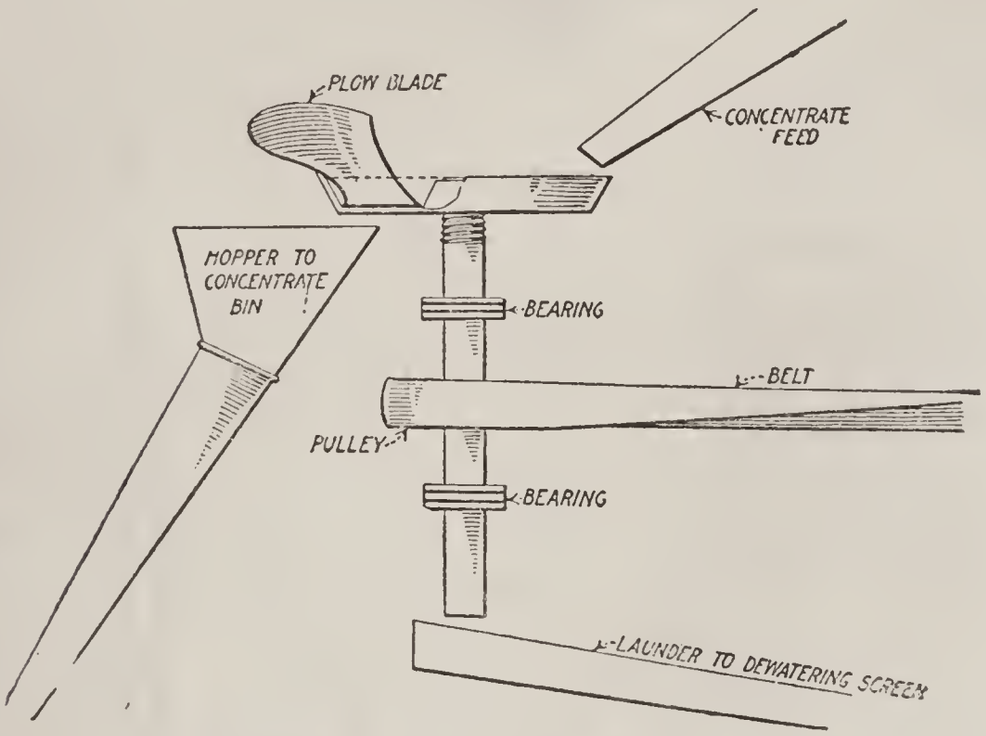
With the introduction of Dorr classifiers in closed circuit with tube mills, the best method of conveying the coarse sand overflow of the Dorr to the tube-mill feed trough became a problem. In recent years the introduction of large spiral scoops on the tube mills has made it possible to wash the sand from the Dorr discharge down the incline to the feed box. But here the question of water regulation enters. Too much water means the rushing of feed material through the mill with no apparent grinding, and too little water threatens us with an overloaded tube mill.

As the maximum efficiency is determined by a certain pulp consistency, it behooves us to get the best regulation of pulp feed. This is obtained with spiral feeders that can be connected to the Dorr-classifier driveshaft by a chain drive. At the discharge end of the spiral feed, the necessary amount of water to keep the proper consistency can be added. With the use of spiral feeders, the frequent shoveling and flushing down of accumulated sands at the end of the Dorr classifier is avoided. The spiral feeders used at the Nevada Packard mill have a life of 7 months, or about 20,000 tons conveyed, with an original cost of about $25. The power cost of the spiral is negligible.

Concentrates Dewatering Pan

A very simple, but effective dewatering pan for mixed concentrates may be put in ahead of the regular dewatering screens, saving much wear and tear.

The device consists of a boiler-plate pan 3 ft. in diameter, made in the shape of a gold pan, with flat



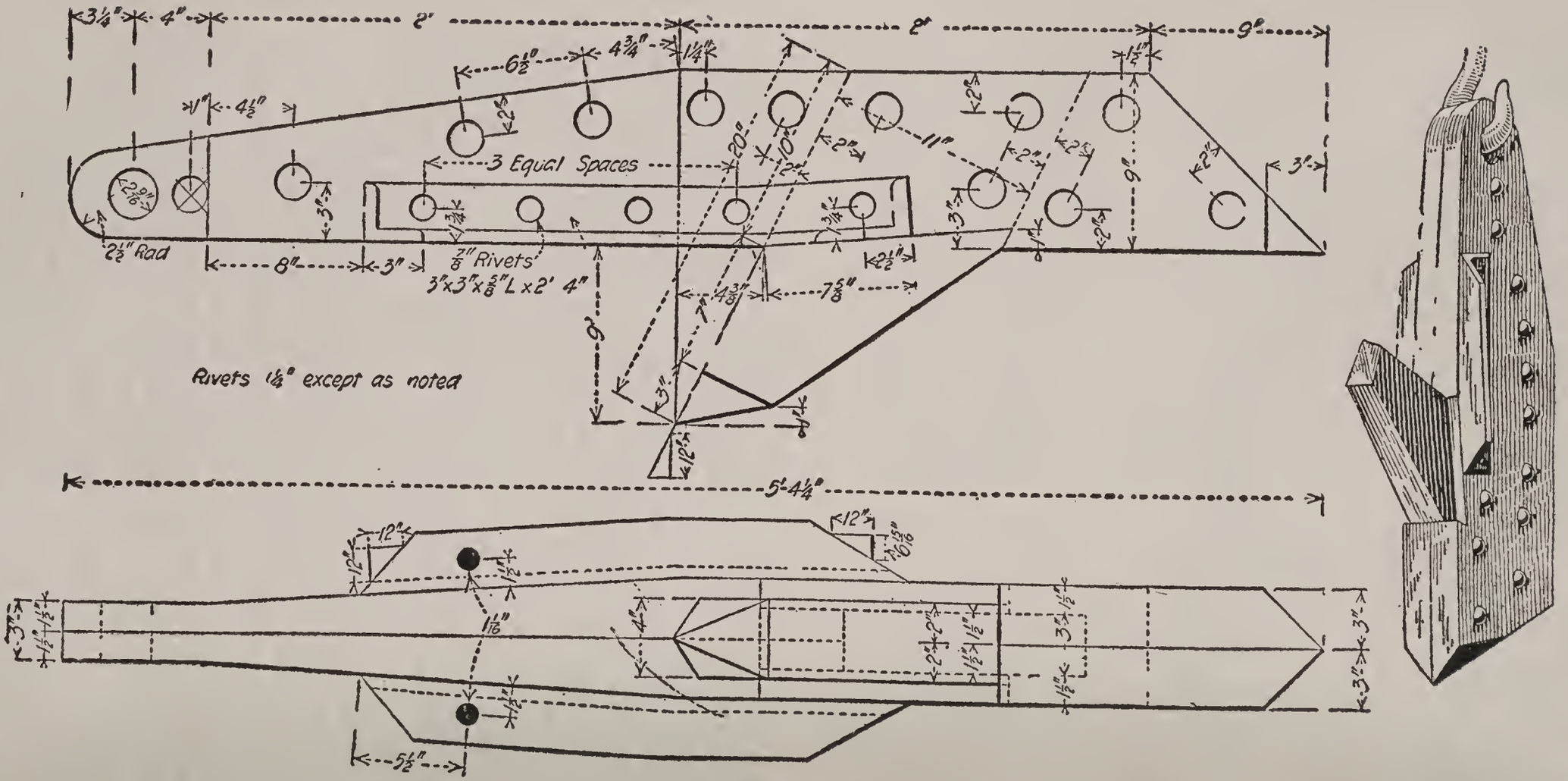
CONCENTRATES DEWATERING PAN

bottom and sloping sides. A central cone 1 ft. in diameter and 4 in. high is fixed in the center of the pan. The top of this cone and the bottom of the pan are bored and tapped for a 4-in. pipe so that the pipe, when threaded, may extend above the top of the cone.

This pipe has a long thread cut on it, and since it is also used as a shaft, it is well to cut out short sections of the thread so that a wedge may be placed between the pipe and the top of the cone to prevent unscrewing or screwing up the pipe, thus changing the height. This pipe is the water discharge from the pan. At some points on the pipe about 1 ft. and 3 ft. below the pan, pipe flanges are keyed and faced to give bearings.

Above the pan an arm is arranged to carry a plow blade, which is made to fit in the pan is to be set above its bottom, the distance depending upon the size of the concentrates. The pipe and pan are then made to revolve, using a rope, a belt or gear drive. The plow blade will remove the concentrates from the pan at an elevation greater than that of the water discharge or overflow. The water will naturally carry some fines, which can be taken to a screen or other dewaterer.

The pan will take all sizes of material from the bull-jig product to table concentrates. The wear and tear on both pan and blade is nominal.



HOOK USED TO REMOVE ACCRETIONS FROM CONVERTER NOSE AT McGILL, NEV.

Assay Conversion Tables

The following table will be found convenient in the case where assay results are reported to the home office in troy ounces per ton (of 2000 lb.) and where separate records must be kept, for official inspection, in grams per 1,000 kg. Most English-speaking mining companies operating in Latin America use the short ton in metallurgical calculations, especially where contents are reported in ounces per ton; and as bullion is usually ultimately sold by the troy ounce, it is only natural that the assay-ton system should be used in calculating.

TROY OUNCES PER TON AND GRAMS PER 1,000 KILOGRAMS[1]

Oz.	Grams	Oz.	Grams	Oz.	Grams	Oz.	Grams	Oz.	Grams
1	34.2859	21	720.0039	41	1405.7219	61	2091.4399	81	2777.1579
2	68.5718	22	754.2898	42	1440.0078	62	2125.7258	82	2811.4438
3	102.8577	23	788.5757	43	1474.2937	63	2160.0117	83	2845.7297
4	137.1463	24	822.8616	44	1508.5796	64	2194.2976	84	2880.0156
5	171.4295	25	857.1475	45	1542.8655	65	2228.5835	85	2914.3015
6	205.7154	26	891.4334	46	1577.1514	66	2262.8694	86	2948.5874
7	240.0013	27	925.7193	47	1611.4373	67	2297.1553	87	2982.8733
8	274.2872	28	960.0052	48	1645.7232	68	2331.4412	88	3017.1592
9	308.5731	29	994.2911	49	1680.0091	69	2365.7271	89	3051.4451
10	342.8590	30	1028.5770	50	1714.2950	70	2400.0130	90	3085.7310
11	377.1449	31	1062.8629	51	1748.5809	71	2434.2989	91	3120.0169
12	411.4308	32	1097.1488	52	1782.8668	72	2468.5848	92	3154.3028
13	445.7167	33	1131.4347	53	1817.1527	73	2502.8707	93	3188.5887
14	480.0026	34	1165.7206	54	1851.4386	74	2537.1566	94	3222.8746
15	514.2885	35	1200.0065	55	1885.7245	75	2571.4425	95	3257.1605
16	548.5744	36	1234.2924	56	1920.0104	76	2605.7284	96	3291.4464
17	572.8603	37	1268.5783	57	1954.2963	77	2640.0143	97	3325.7323
18	607.1462	38	1302.8642	58	1988.5822	78	2674.3002	98	3360.0182
19	641.4321	39	1337.1501	59	2022.8681	79	2708.5861	99	3394.3041
20	685.7180	40	1371.4360	60	2057.1540	80	2742.8720	100	3428.5900

[1]Equivalents of the fractional parts of an ounce (per ton) in grams per 1000 kg. are as follows: 0.1 oz., 3.4286; 0.2 oz., 6.8572; 0.3 oz., 10.2858; 0.4 oz., 13.7144; 0.5 oz., 17.1429; 0.6 oz., 20.5715; 0.7 oz., 24.0001; 0.8 oz., 27.4287; 0.9 oz., 30.8573.

Converter-Nose Hook

To remove the collars or accretions that form around the nose or snout of a copper converter, various types of hooks have been used. With the customary form of bent hook, difficulty is often experienced in securing a "bite" and finally the hook becomes heated and bends, losing its hold. The particular advantage of the hook employed at the Steptoe plant of the Nevada Consolidated Copper Co., at McGill, Nev., and shown in the accompanying illustration, is its great strength due to the fact that the grain of the metal in both shank and barb is straight and unstrained. This device is practically the same as that developed by Thomas Taylor at the smeltery of the United Verde Copper Co. at Jerome, Ariz. At the Phelps-Dodge works in Arizona, some hooks involving a similar principle have been adopted. These hooks, however, have the "biting" bar pivoted in the vertical element with one leg of the bar longer than the other; this causes the short end, in which is inserted a special steel tool, to assume a "biting" angle, while the long leg assists in lifting the collar.

Laboratory Efficiency

Time-saving devices in the assay laboratory are very valuable, as there are so many processes involved that a minute saved here and there soon amounts to hours.

There are just so many steps to be performed, so the question resolves itself into: How can the time of these individual operations be cut down? In all laboratory work, there is a certain routine that always exists, and one is likely to get into the rut and remain there unless one is wide-awake and continually on the lookout for short methods and time-saving devices.

Some, and perhaps all, of the hints given here may be known to many, but let us hope that someone will find one at least that he has not heard of or used and thereby may be benefited.

Sheets of thin wrapping paper about 9 in. square used in lieu of the small sample pan for pulps ready for mixing and weighing. They are easily handled, eliminating all chance of salting as they will not permit of

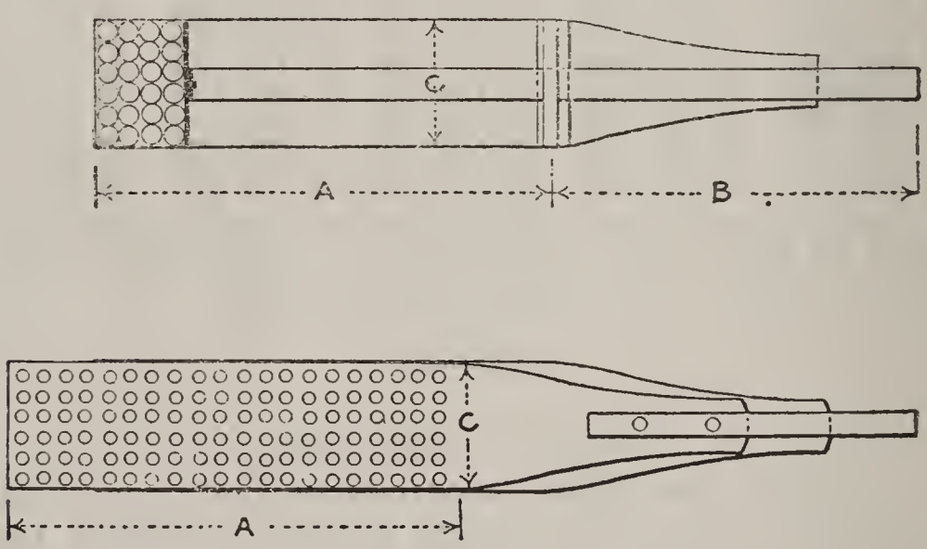


FIG. 1. LOADING TRAY FOR CUPELS

stacking, though a little more bench room may be required.

A common bar shaker consisting of a glass and an aluminum receptacle is useful for the thorough mixing of pulp. The mixing of all samples can be done at one time where papers are used instead of pans, and then the samples are ready for weighing. Mixing thus is easier than using a cloth, is more uniform and is done in about half the time. After the weighing operation has been completed, the shaker is again used in mixing the pulp and flux, instead of the spatula. Much time can be saved here also.

Loading trays for placing cupels in the muffle and buttons in the hot cupels. Figs. 1 and 2 show these. They are made of sheet iron of a thickness depending upon the furnace used; that is, ordinary cupelling

muffles only will permit of a thinner metal than the hot crucible furnace.

The first tray (see Fig. 1) is so constructed that its length *A* will be the length of the muffle, its width *C* such that it will just take a row of cupels packed tightly together to avoid rearranging them when they are placed in the muffle. In this way a full muffle may be loaded in an instant and the same with the lead buttons. *B* is of arbitrary length, about 12 in. being very convenient.

Considerable care should be exercised making the second tray. First, the holes should be exactly centered, and then they should be drilled or punched from the under side and the edges allowed to project quite a little way so that the lead buttons will not be pushed out of place while loading them into the hot cupels. When the tray is loaded and ready for the furnace, place it carefully over the cupels in the muffle, pull out the lower

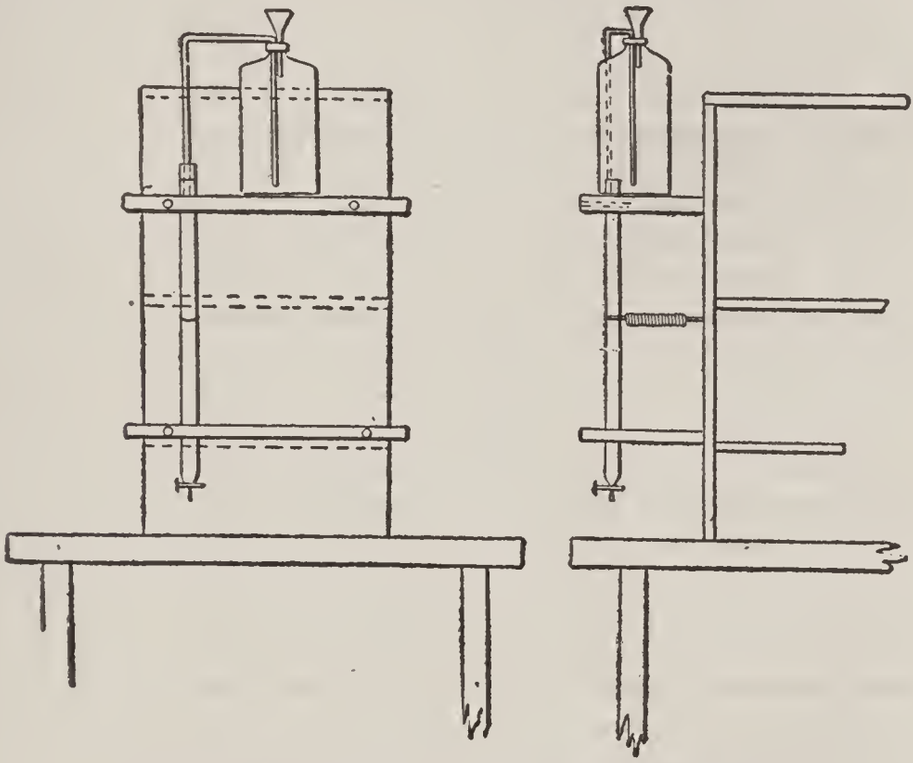


FIG. 2. A SIPHON FOR PARTING ACID

plate and all of the buttons will drop into their own cupels. This is one of the best time savers that I have ever used in my laboratory, not to consider the intense heat that is thus avoided.

Referring to Fig. 2, which is a continuous siphon used in filling parting cups with acid, the sketch is sufficient to explain the apparatus. Any reservoir can be used, but as rubber tubing must be avoided, a one-way burette is the most convenient.

A sheet-metal tray for parting cups with sides at such a height that when the tray is placed on the hot plate, all the cups will rest on the plate, eliminating the necessity of their removal. Annealing may be performed in the same tray in case one desires to anneal in the furnace, though personally I prefer the use of a small hot-blast lamp as the life of the cups is much greater.

Pennsylvania Gold Mining Co. Form No. 1

MILL REPORT

Nov. 25, 1916.

BATTERY NO.	6	5	4	3	2	1	REMARKS
Time run (hours)	7	7 ¾	6	7 ¾	4	4	
Hung Up (hours)	1	¼	2	¼	4	4	
Reason hung up							
Brushing	×	×	×	×	×	×	
New Screens			× × ×				
Shoes			× ×				
Dies			×				
Ore Short					×	×	
Fixed broken stem	×						

Quicksilver fed battery 18 Oz.

Quicksilver on plates 3 Oz.

Amalgam from plates 66 Oz.

Requisitions *broom, belt 16″x48′—5 ply.*

Supplies received 50 *ft. hose*—481 *oz. quick.*

Labor *J. Jenks, rockbreaker* 8, *Jackson, repair* 2.

J Boyd Millman

Stamp-Mill Reports

One of the difficulties of preparing blank forms for mill, mine or other labor reports is the composing of something easily understood, which the workmen can readily and honestly fill in from the data encountered in his daily routine. Any blank form that will bring before his eyes the work he has performed during the day will be more likely to interest him and secure his co-operation than one so intricately planned or worded that he has to guess at the answer.

After trying various types of stamp-mill reports, with more or less success, the one reproduced herewith was composed. The idea incorporated in it was conceived while watching a dentist mark his chart indicating the treatment of a patient's teeth, hence the application of the idea is believed to be original.

The chart represents a 30-stamp mill as one stands in front of the stamps. Each battery has a number, and the chart column representing it is divided into five spaces, each representing a stamp. The usual repairs, changes or renewals are printed at the left, blank places being left for reporting unusual conditions, which should be written in when they occur. It is a simple matter for the millman to put a × in the column representing a certain battery or stamp, opposite the given repair, change or renewal. Spaces at the bottom of the report show the amount of "quick" fed, amalgam removed, and requisitions desired or filled, and another space is provided for reporting labor employed in the mill.

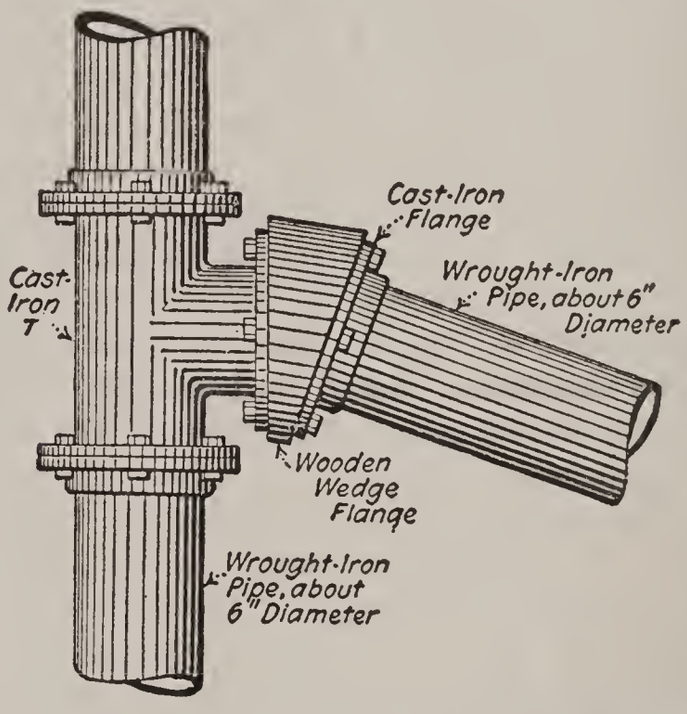
When these reports reach the office and are properly filed, it is an easy task for the clerk or superintendent to scan them quickly and catch the × showing any renewal or repair in question. One knows to the exact stamp and date just the information wanted without having to laboriously read a lot of matter foreign to the question. These charts are found a fine supplement to a tabulation. One cannot peruse them without getting near to the millman and his real need. They keep one informed better than a mere tabulation of figures from which all human feeling has been obliterated. In the use of this report the millman will not leave the manager in ignorance of his wants, for he is sure to stick some note on the margin, telling something perhaps previously unknown and many times giving the clue to the trouble the manager has been looking for.

This graphic report is easily tabulated for monthly reports, and may be worked out in many different ways. The report reproduced herewith indicates rather more than the usual ills of one shift in a mill, but is so given to more clearly illustrate the method. The chart shows

that batteries Nos. 3 and 5 ran the regular time without trouble, only stopping to brush up the plates. Batteries Nos. 1 and 2 lost four hours each because of shortage of ore. Battery No. 4 had trouble with the shoe and die of the feed stem and the adjoining shoe on the second stem, and took advantage of the hang-up to change the screen. If new shoes or dies had been put in, the word "new" would have been put before the word shoes or dies. Battery No. 6 had a broken feed stem and lost an hour fixing it. The rest of the report is obvious.

A Wooden Angle Flange

In the concentrating mill of the Old Dominion at Globe, Ariz., an ingenious method of making a pipe connection at an angle without the use of special fittings, which cost money and require time, was devised as

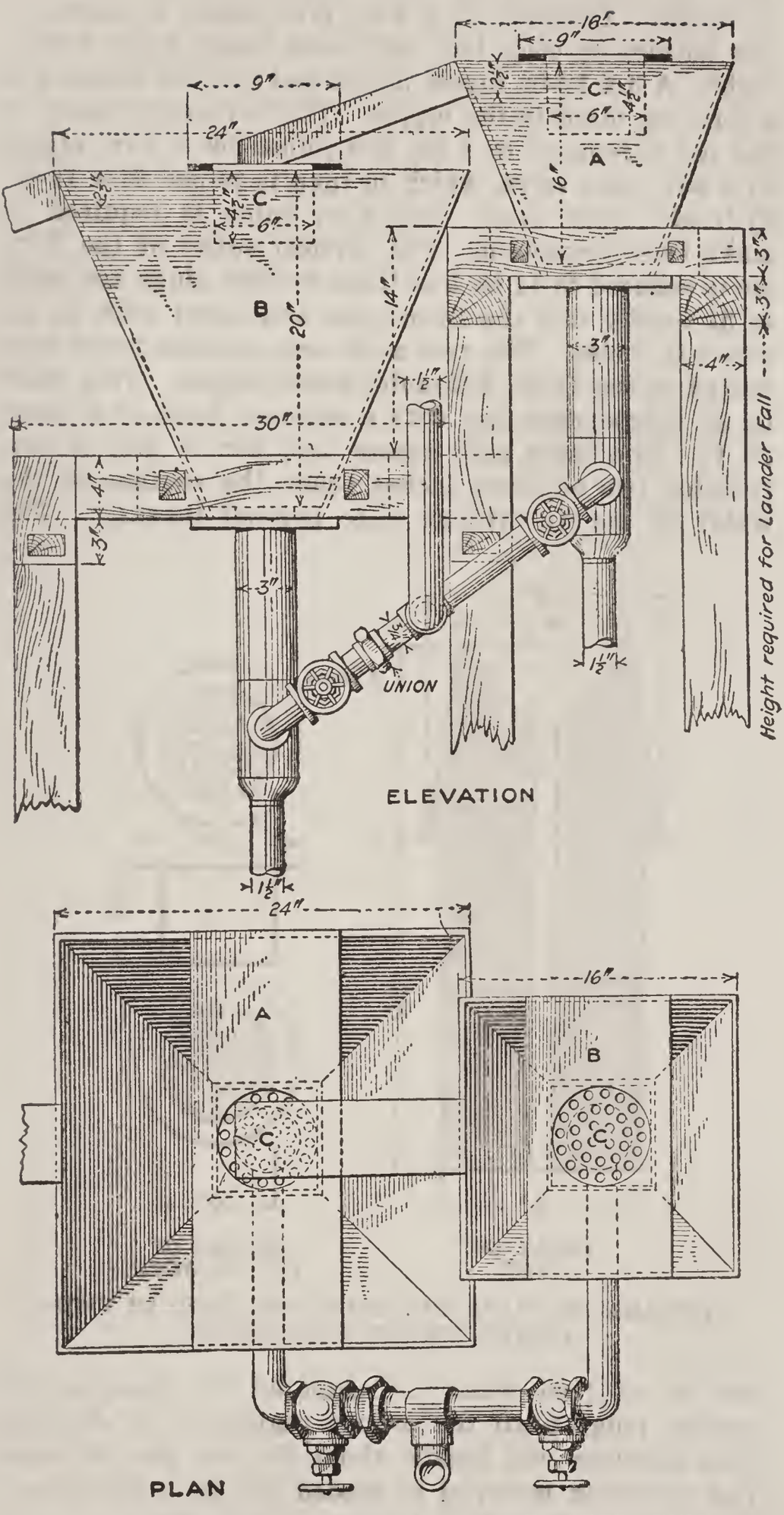


A WOODEN ODD-ANGLE PIPE FITTING

shown in the sketch. By simply cutting a wooden wedge to fit snugly between the flanges, boring and trimming the outside and using the customary bolts and gaskets, a perfectly tight and satisfactory job was secured.

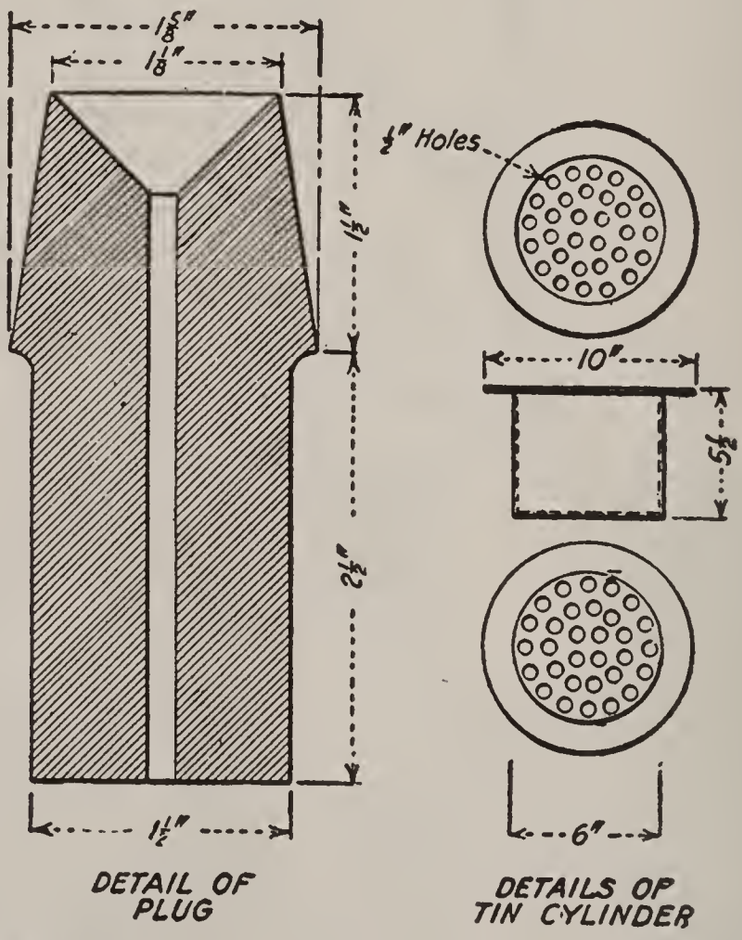
Making an Inexpensive Classifier

In isolated localities or in small mills for working tailings dumps a small two- or three-unit classifier is often essential. The following is a description of a two-compartment classifier that can be made by any mill carpenter and gives excellent results.



DETAILS OF A SIMPLY CONSTRUCTED TWO-COMPARTMENT CLASSIFIER

The two compartments, A and B, are wooden boxes built in the form of truncated pyramids of dimensions as shown. One-half of a 3-in. pipe flange is bolted to the bottom of each box, all joints being made watertight. A 3 x 12-in. nipple is screwed into the flange and a 3-in. tee fitted to the nipple. The horizontal outlet of the tee is reduced to ¾ in. and joined by a ¾-in. nipple to a ¾-in. gate valve, which in turn is joined to a 1½-in. hydraulic water line. One ¾-in. union is required to make this connection. The vertical outlet of the 3-in. tee is reduced to 1½ in. and three wooden plugs are made to fit tightly into the 1½-in. pipe and bored with ⅜-, ½-, and ⅝-in. holes. The two units are assembled and connected to the 1½-in. hydraulic-water supply. This must be an independent line with a constant head of at least 20 ft. The table pulp passes into box A and is distributed two or three inches below the surface of the water by the tin cylinder C to prevent foaming. The



DETAILS OF PLUG AND CYLINDER USED IN TWO-COMPARTMENT CLASSIFIER

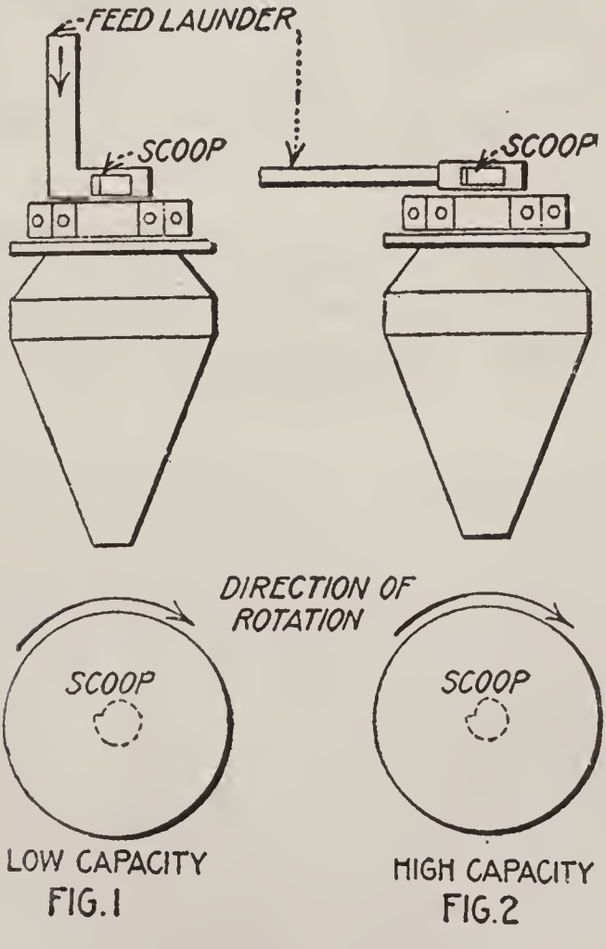
amount of feed water is regulated by changing the wooden plugs until the proper regulation is obtained. This classifier will handle about 50 tons per 24 hours. The following material is needed for its construction: 20 ft. bd.ft. T and G lumber, ⅝-in.; two 3 x 12-in. nipples; two 3-in. tees; two ¾-in. gate valves; one ¾-in. union; seven ¾-in. nipples; two ¾-in. 90-deg. ells; two 1½-in. short nipples; one 3-in. flange union; two sets

of reducers 3 in. to ¾ in.; two bushings ¾ in. to 1½ in.; one 1½-in. tee; sufficient 1½-in. pipe for hydraulic-water supply line, and 4 x 4-in. lumber for framing to suit requirements; also two tin cylinders, which may be made from any suitable material available.

Additional compartments can readily be added to the classifier if desired. Only one hydraulic line and union will be needed for any number of compartments (provided there is sufficient head). It is convenient to mark the wheels of the valves at the ¼, ½ and ¾ points.

Increasing Ball Mill Capacity

In a Western milling plant where a Hardinge ball mill was used to reduce the size of ore after coarse rolls, it became necessary to install a second mill to increase the grinding capacity. The feed to the first mill entered the scoop-box at right angles to the direction of rotation of the mill, as shown in Fig. 1. When the second unit was installed, owing to the restricted floor space available, the feed launder conducted the feed to the scoop-box as shown in Fig. 2. It immediately became apparent that the grinding capacity of the plant was considerably more than doubled, and upon ore of medium hardness the second mill alone had sufficient capacity to supply the rest of the plant. As



FIGS. 1 AND 2. SHOWING INCREASED CAPACITY WITH CHANGE OF LAUNDER

the two machines were identical in all other respects and each received the same kind of feed, it was evident that the difference in grinding capacity was due to the difference in direction of feed-flow to the scoops. The feed to the second mill flowed into the box at a considerable velocity and straight into the scoop mouth as it moved on that portion of its revolution through the feed, consequently more ore was actually forced into this mill than into the first, and a greater grinding capacity was the result.

Of course there would necessarily be a limit to the increase in grinding capacity of the mill with increase of feed, but the indisputable fact that such an increase took place in this instance suggests that possibly, on account of insufficient crowding, some other mills are not operating at capacity and that perhaps some experiments along this line would be profitable.

Interesting Belt-Cutting Kink

In spite of all the belt kinks I have read and the number of times belt men have been cautioned to "cut the belt square" before lacing, here is a good one that insures straight running even though the cut is not made square. In fact, a straight-edge or steel square

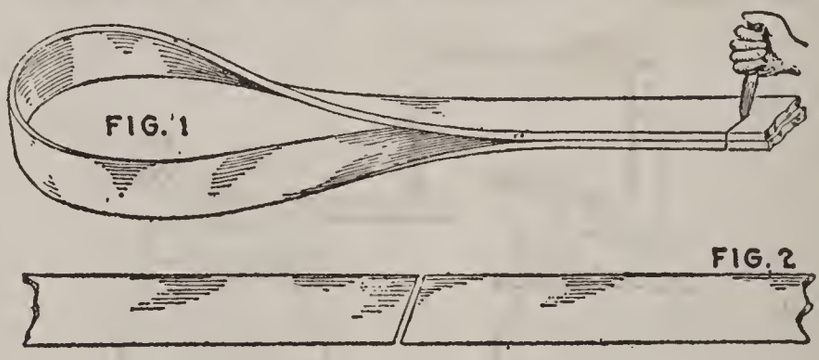


FIG. 1. SHOWS HOW THE BELT IS FOLDED, FIG. 2 HOW IT MATCHES

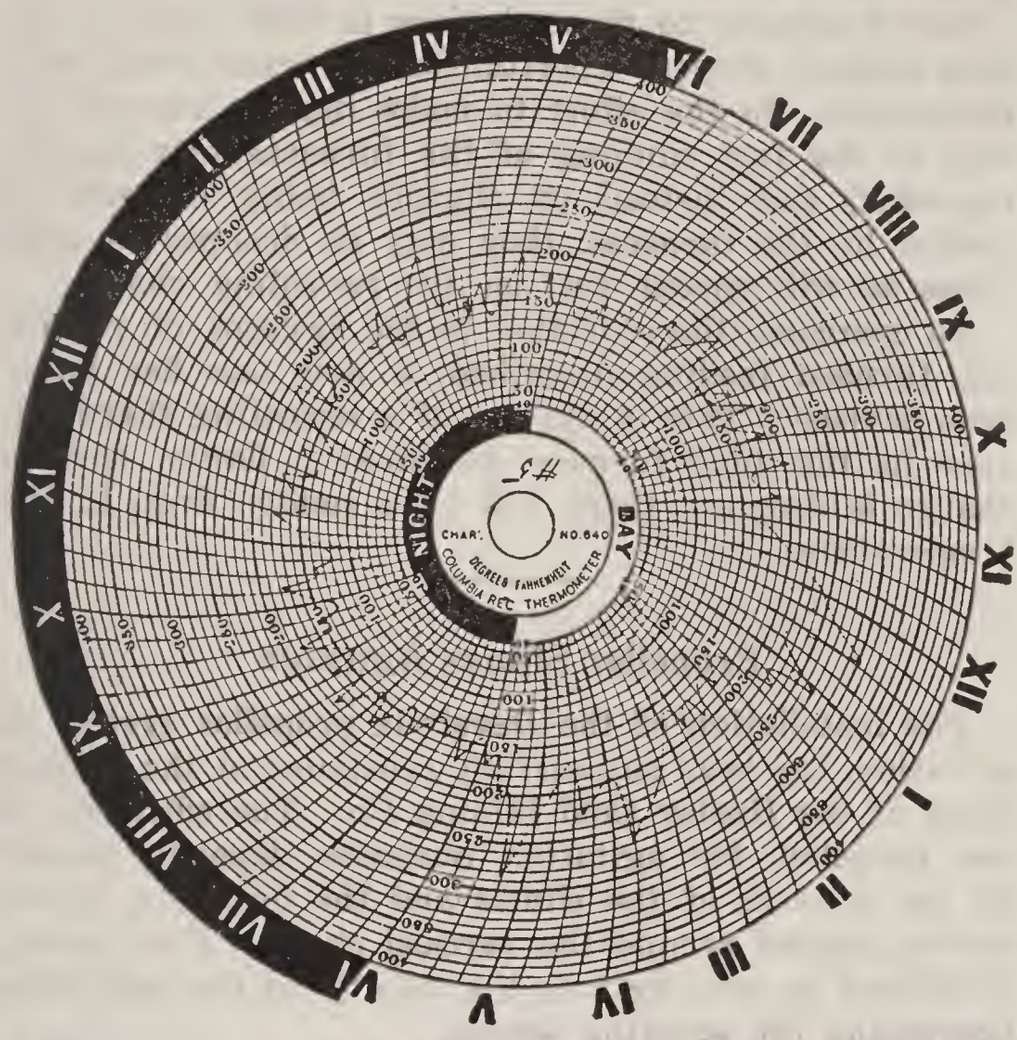
is not needed at all for jointing narrow belts. J. C. Conn is the discoverer of the method.

The sketches show the method plainly—Fig. 1 how the belt is laid for cutting and Fig. 2 how the ends match. Match the two ends, one on top of the other as shown, with both smooth sides either up or down and the sides perfectly even. Then cut both together along a straight line. If cut square, at 45 deg. or any other angle, the joint will be a perfect fit.

In case of double or triple belts, where both sides are smooth, place the "outside" of the belt either up or down. The belt must be twisted so that *the same side is on top* at the point of cutting. The reason is evident.

The Regulation of Blast Furnace Feeding

In the feeding of practically all types of blast furnaces regularity is a *sine qua non* if good work is to be obtained. This is particularly true of silver-lead furnaces. There is considerable temptation for feeders to fill up furnaces quite high and then sit or lie down for a



A TEMPERATURE RECORD SHOWING THE VARIATIONS CAUSED BY FEEDING

good long rest. In order to prevent the irregularities resulting from this practice, many expedients have been resorted to by various superintendents.

Perhaps as common a means of watching the furnace work as any is the observation of the blower revolutions when the blowers are steam-driven and pressure is maintained by speed variations. If furnaces are allowed to get unduly low, the blowers will speed up, and vice versa. Hourly or half-hourly records of blower revolutions are often enlightening as to what is going on at the furnaces. As a measure of the regularity of feeding, however, they are not quite satisfactory, for a change in the character of the charge from coarse to fine will slow down the blowers, and so on. Knowing that the charge columns are maintained at the correct heights, the blower revolutions are then really indicators of the character of charge being fed.

When the charge cars are electrically propelled, a

recording ammeter on the tramway circuit will show just how frequently charges were delivered. Electrical devices have been used that make marks on revolving charts every time charges are dropped. One of these on each furnace is better than the ammeter on the whole circuit, as with such an installation each individual furnace can be kept track of.

One drawback to these devices is that they do not give feeders proper opportunities for exercising their judgments. In the effort to make regular records the slow or the fast running of the furnaces may be disregarded, and instead of charges being dropped as needed by the furnaces, they may be dropped at such times as will appear well on the recording charts.

To overcome the objection to the systems of recording furnace feeding described, a new one has lately been devised. This is the use of a recording thermometer in the blast-furnace downtake. Every time a charge is dropped, there will be a sudden drop in the temperature.

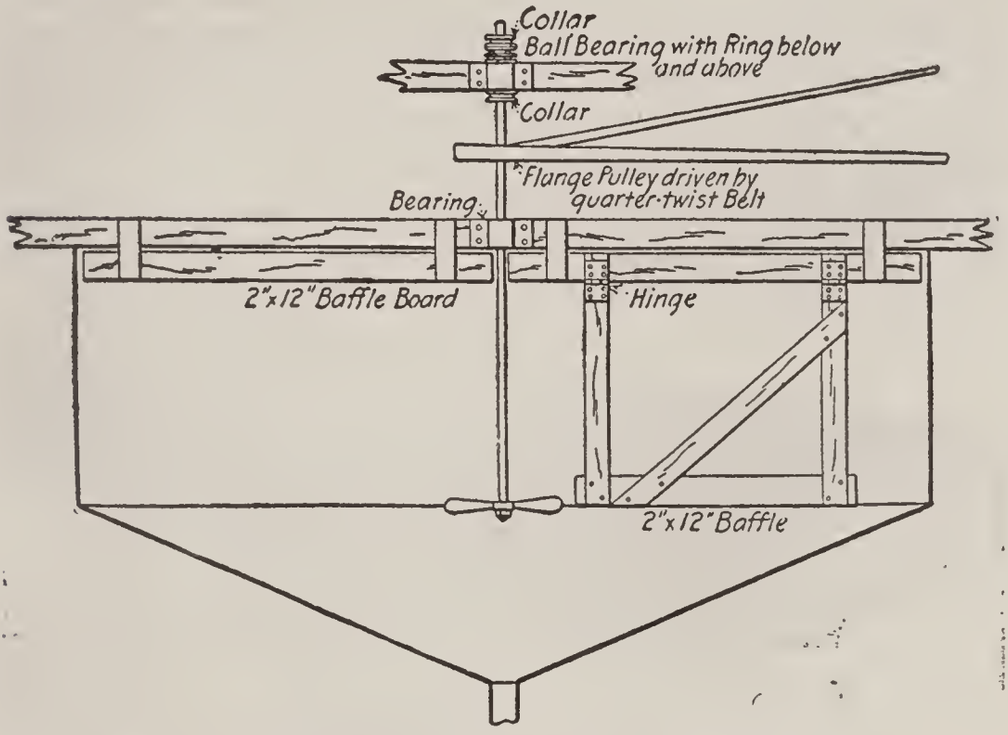
Propeller Slimes Agitator

The installation of the Devereux propeller system of agitation, has been successfully continued at the milling plant of the Hedley Gold Mining Co. The efficiency of the propeller for agitation purposes depends largely on the fineness of the slimes and the breaking of the vortex formed around the vertical shaft. This vortex, if allowed to exist, retards the velocity of the pulp, thus decreasing the scouring action.

The vortex is broken by baffle boards 2 in. thick by 12 in. deep suspended from the lower chord of the truss and extending from a few inches from each side of the vertical shaft to within 3 in. of the inner side of the tank. To allow agitation while the tank is filling and to prevent the pulp from surging, another baffle board is suspended by hinges from one of the former baffle boards and on one side only of the vertical shaft. The rope holding this baffle in vertical position while tank is filling is slackened as soon as the tank is full, thus allowing this baffle to lie flat on the surface of the charge and not interfere with the scouring action below. From Trautwine, page 577, a velocity of $\frac{1}{4}$ ft. per second will wear away ooze and mud; $\frac{1}{2}$ ft. per second will wear away clay; 1 ft. per second will wear away sand.

With the 5-ft. propeller revolving from 80 to 100 r.p.m., maximum hp. = 12, the 34-ft. diameter, conical-bottomed tank has handled from 80 to 165 tons of dry slimes. Then 138 tons of dry slimes plus 296 tons of

solution, or 31.8 per cent of solids to 68.2 per cent of solution, is an ordinary charge and is agitated thoroughly with 80 r.p.m. The 4-ft. propeller revolving 80 to 100 r.p.m. maximum hp. = 10, in the 30-ft. diameter, conical-bottomed tank will handle 138 dry tons of slimes (41.4 per cent) plus 238 tons of solution (58.2 per cent) when revolving 80 r.p.m. Either tank will handle slime charges with any proportion of solution used in ordinary cyanide work, and the power



THE PROPELLER SLIMES AGITATOR

does not exceed that stated. There is no deposit of slimes alongside the staves, the whole charge being in rapid motion.

As extraction is effected by a series of agitations and decantations, one might expect trouble in starting. Thus far none whatever has occurred, even after two days' time between agitations. With the throwing in of the handle of the motor starter, the propeller begins setting the charge in motion, and by the time the tank is full all slime is moving rapidly.

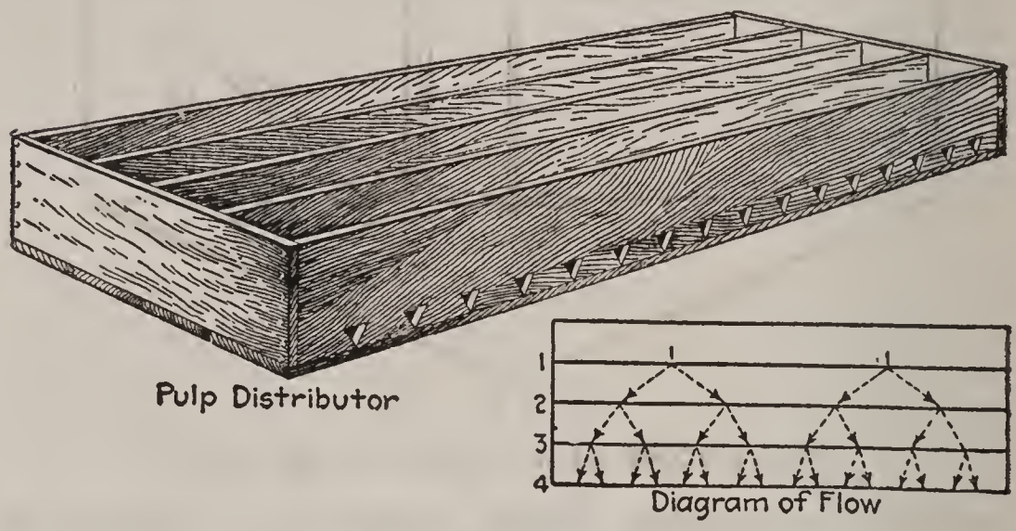
The cost of these propellers is said to be less than half that of any other agitators going into new as well as remodeled plants. The system is dependent upon power from one source only—air is not needed. Its simplicity and adaptability permit it to be installed in plants containing tanks of any style, height or diameter.

An Efficient Pulp Distributor

In plants where plate amalgamation is used for recovery of gold, it is important that the pulp be distributed evenly over the surface of the plates. With the splash from Chilean and Huntington mills this is not as easily

accomplished as it is in the case of stamps. In amalgamating below such machines, I have used several forms of distributors, but have obtained by far the best results with the one shown in the accompanying sketch. It can be made in about two hours by anyone having any carpentering ability. For use at the head of the average-width plate the box will need to be 4 ft. long, 1 ft. wide and 4 in. deep, and made of 1-in. surfaced lumber.

Referring to the diagram, the partition marked 1 contains two triangular openings with the points down, the total area of the two openings being equal to the area of a 2-in. pipe. The next partition has 4 triangular openings the total area of which equals that of the preceding two. Partition 3 contains 8 triangular openings the total area of which is equal to the 4 in the



A GOOD PULP DISTRIBUTOR

preceding partition. Partition 4, the final discharge, contains 16 openings the total area equivalent to that of the 8 preceding ones.

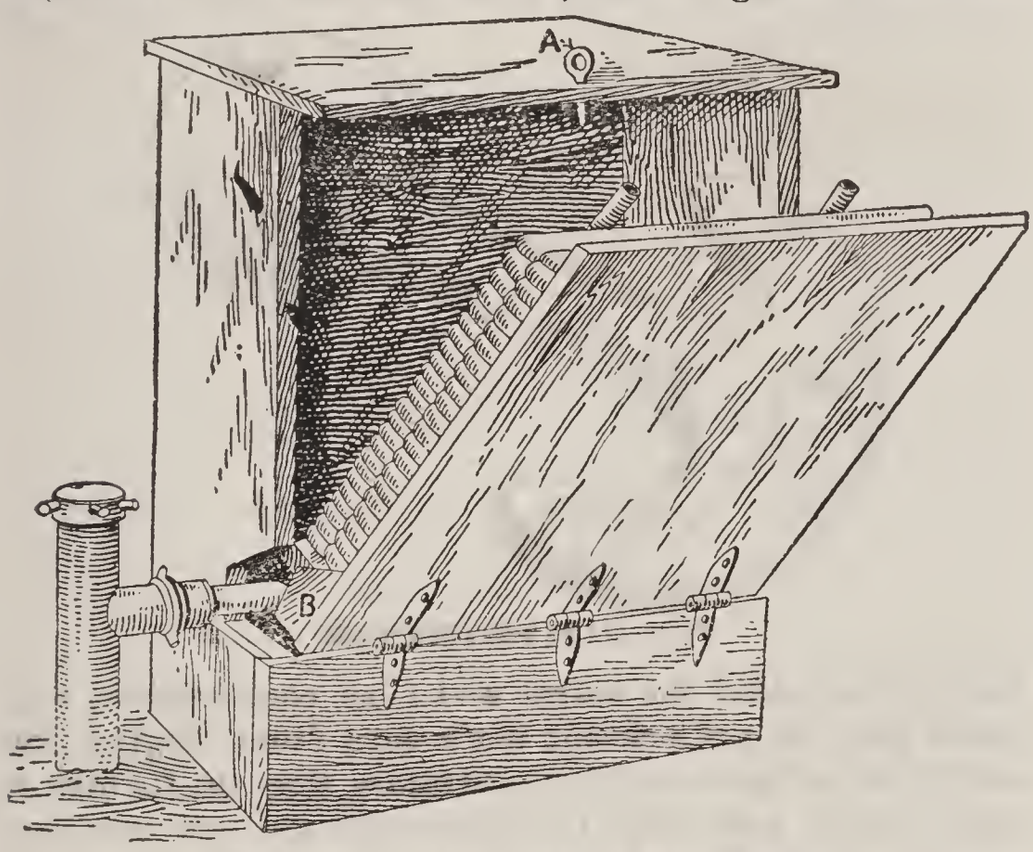
Each of the openings must be accurately spaced, so that the flow from it is evenly split and feeds two of the succeeding openings. If the mill pulp is delivered in the middle of the first partition, the box perfectly level lengthwise and tilted with the pitch of the plates, the tailings will "hutch" about each of the series of openings and an absolutely uniform flow will be obtained from each of the discharge openings. Round or square holes will not give the results obtainable from the triangular-shaped ones. I make no claim for the originality of this device, but have used it with success in a number of mills.

Outside Rack for Fire Hose

Fire protection for industrial plants is an important branch of safety engineering. Adequate fire protection cannot be had without proper care and easy accessi-

bility of fire-fighting apparatus. The illustration shows a serviceable and convenient hose rack for mills, smelteries and mine plants. It consists of a wooden frame about 4 ft. high, 5 ft. long and 12 in. wide inside, covered with corrugated iron, preferably galvanized, with top hinged in the rear and front hinged at the bottom, the two doors, or lids, being held together by the pin *A*. At the bottom of the front door is a shelf so fastened to it that the two move together. The rack is placed with one end about a foot from the fire plug, and at the bottom of that end a piece is cut out, as shown; the triangular portion *B* of this piece is fastened to the front door.

The hose is piled upon the shelf and the plug end is brought out through the rectangular opening at the bottom of the rack. In case of fire it is only a few seconds' work to pull out the pin *A*, allowing the front door to fall forward, and drag the nozzle end



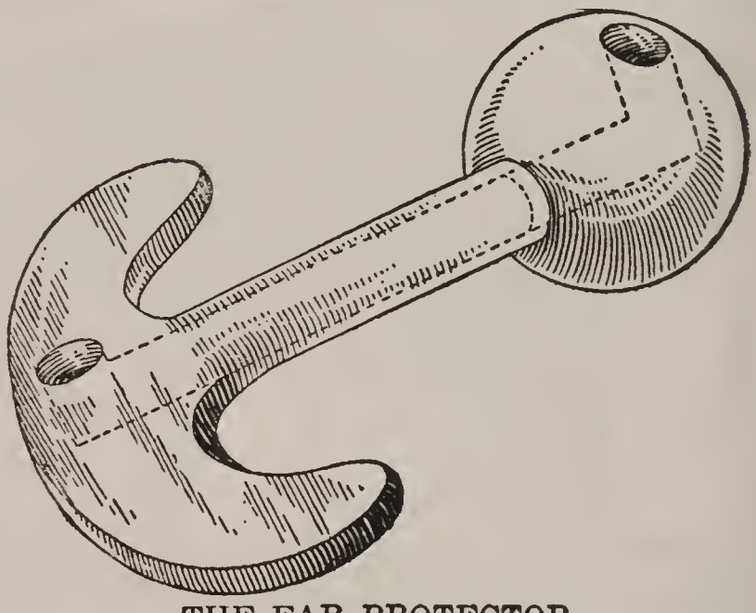
OUTSIDE RACK FOR FIRE HOSE

of the hose to the fire. The triangular opening in the end of the housing gives plenty of clearance for the hose, which, carefully piled upon the shelf, pays out readily. As the hose is always fastened to the plug, there is no time lost in making connections and water can be on the fire as soon as the nozzle men are ready. The short piece of hose exposed between housing and plug can be protected by an extra box placed over it. This rack is inexpensive, takes up little room and answers every purpose. The device is obviously simple to construct and furnishes a means of protecting the hose from dust and weather conditions, while providing immediate readiness when necessary.

Ear Protector for Millmen

Those of us who have had to work for any length of time in the noise of the stamp mill have seen the ill effects of the deafening roar, and the curious phases of the adjustment of the ear to its unusual surroundings. Many millmen, who have become quite deaf to ordinary conversation in the open air, can hear quite distinctly amid the roar of the stamps, if the voice is only slightly raised and the speaker is quite close. They can also differentiate between the usual crash and pound of the stamps and the slightly different noises resulting from too close feeding of the stamps or a loose stem. The ear adjusts itself to the vibration, and is rather uncomfortable when away from it.

Many millmen protect their ears by stuffing them full of cotton and soap, or even waste. For a number of years I have used the ordinary naval big gun ear protectors, when working in a mill, and find the form



THE EAR PROTECTOR

herein illustrated the safest and most comfortable; and while they do not prevent one's ears from ringing for awhile after removing the protectors they do furnish a great relief, especially for those who are not continuously in the mill, and whose ears therefore do not get adjusted to the vibration.

The protector is made of celluloid, and consists essentially of a ball containing a round hole on one side, and a small duct leading from the bottom of the hole through the stem to another smaller hole in the anchor shaped terminus of the stem. The ball is inserted in the ear, the hole next to the ear drum, and the anchor shaped stem is braced against the lobe of the ear, thus holding the protector in place. The sound now has to enter the small hole in the stem, traverse the narrow passage or duct to the hole in the ball, and thence pass to the drum of the ear. The sound therefore has to pass through a decreased aperture and make two sharp

bends before reaching the ear; the effect is to reduce the shock and vibration to the ear drum without preventing one from hearing entirely.

There is nothing in the shape or method of insertion of this protector to injure the ear in any way, which is more than can be said of some of the protectors or remedies adopted by millmen.

The Assaying of Copper Bullion

In the autumn of 1913, owing to revolutionary disturbances, a large part of the copper ores coming to the Aguascalientes smeltery was cut off, and at the same time the receipts of gold and silver ores from southern Mexico were greatly increased. In consequence the blister copper produced began to carry abnormal amounts of precious metals. The method of assay in use at the time was the "combination" nitric acid for silver and "all scorification" for gold. This proved unsatisfactory under changed conditions, so I did some experimenting with the sulphuric-acid method in use at Anaconda, finally adopting it altogether in preference to the other.

Four charges of 15 grams each are weighed into 800-c.c. beakers; about 30 c.c. of water is added to each, and then 10 c.c. of mercuric nitrate solution (40 grams of mercuric nitrate per liter). This is shaken till the copper is coated with mercuric amalgam, then 60 c.c. of con. sulphuric is added, the beaker covered and put on hot-plate. Action begins at once with evolution of SO_2 gas. Heating is continued for about 1½ hr. till the solution becomes bluish-gray. I find the best results are obtained by starting the action at a rather high heat.

When the action is finished, as shown by the bluish-gray color, the beakers are removed from the hot-plate and allowed to cool. When cold, the sides are washed down and sufficient salt solution added to precipitate the silver as chloride (the greater part of the silver is reduced to metallic state by the copper, but salt solution is always added). The solution is then brought to a boil and, when possible, allowed to stand over night. It is then filtered, washed a couple of times and the paper and contents transferred to a 20-gram crucible. The paper is burned at a low heat, flux added and the contents fused. The resulting buttons are cupelled, weighed and parted as usual.

The comparative results on fifteen lots are given in the table.

ASSAYS GIVEN IN KILOGRAMS PER METRIC TON

Lot No.	Au by Scorification	Au by H_2SO_4 Combination	Ag by HNO_3 Combination	Ag by H_2SO_4 Combination
1	0.2820	0.2817	33.307	33.342
2	0.3821	0.3814	45.740	45.734
3	0.3733	0.3739	49.053	49.072
4	0.2806	0.2861	37.335	37.332
5	0.2920	0.2927	34.846	34.972
6	0.2762	0.2778	29.992	30.083
7	0.2383	0.2381	24.995	25.017
8	0.2283	0.2287	23.930	23.913
9	0.2211	0.2221	22.618	22.689
10	0.1935	0.1928	20.780	20.680
11	0.1363	0.1386	19.272	19.334
12	0.1368	0.1363	19.412	19.384
13	0.1432	0.1446	18.925	18.901
14	0.1964	0.1956	31.818	31.869
15	0.3450	0.3465	40.988	40.984
Average.....	0.2483	0.2491	30.200	30.221

As the sulphuric-acid method gave good results, the other was discontinued and no further comparisons were made. It may be of interest that the values in the bullion increased to a high mark of 88 kg. of silver and 1,200 kg. of gold.

The combination nitric-acid and scorification method cost 84c., while the sulphuric-acid method cost 29c. per assay for materials only.

The Analysis of Tin Ores

The increasing importance of tin has rendered necessary a more rapid and accurate method for the analysis of tin ores than has heretofore been available. The principal methods in use are the fire assay with KCN and Low's iodometric method. Any fire-assay process is obviously subject to large losses, and cassiterite ores are not readily decomposed by Low's method. The present method, worked out through the assistance of Prof. V. H. Gottschalk and R. S. Dean, is based upon the titration of the sulphide with potassium iodate, as has been described by Mr. Dean for other sulphides.

The ore, whether cassiterite or sulphide, is decomposed by fusion with potassium carbonate and sulphur (1 part S to 1 part K_2CO_3), which we found may be most conveniently done in a Jena Erlenmeyer. About 45 min. is usually sufficient for complete decomposition of the ore. The melt is then treated with warm water and filtered, the tin passing into the filtrate as potassium sulpho-stannate. This filtrate is then evaporated and taken to acid sulphate melt, also most conveniently made in an Erlenmeyer according to the directions given by Low.

This fusion is taken up with 1:1 HCl and treated with H_2O_2 to insure complete oxidation of the tin to the stannic state. The tin is then precipitated as SnS_2 by H_2S. This precipitate is filtered and washed free of H_2S. The sulphide, together with the filter paper, is

then placed in a known amount of standard iodate and hydrochloric acid added so that the solution is 30 per cent by volume strong HCl (sp.gr. 1.2) measured at the end of titration. The titration is then finished with iodate if an excess has not been added, or if an excess has been added it is titrated back with standard iodide solution using CCl_4 or $CHCl_3$ as indicator in either case. Some of the results obtained by this method are shown in the following table:

RESULTS OF ANALYSES OF TIN ORES

Pure Sn	C.C. Used	0.05 Molar Iodate, Theoretical
0.1	16.8	16.82
0.1	16.8	16.82
0.1	16.85	16.82
0.1	16.8	16.82
0.1	16.7	16.82
0.1	16.9	16.82
Sn. ore fire assay, 7.92%; iodate		72.7 72.85

By this method the sulphide is oxidized entirely to sulphur according to the equation,

$$SnS_2 + KIO_3 + 6HCl = SnCl_4 + S_2 + KCl + ICl + 3H_2O$$

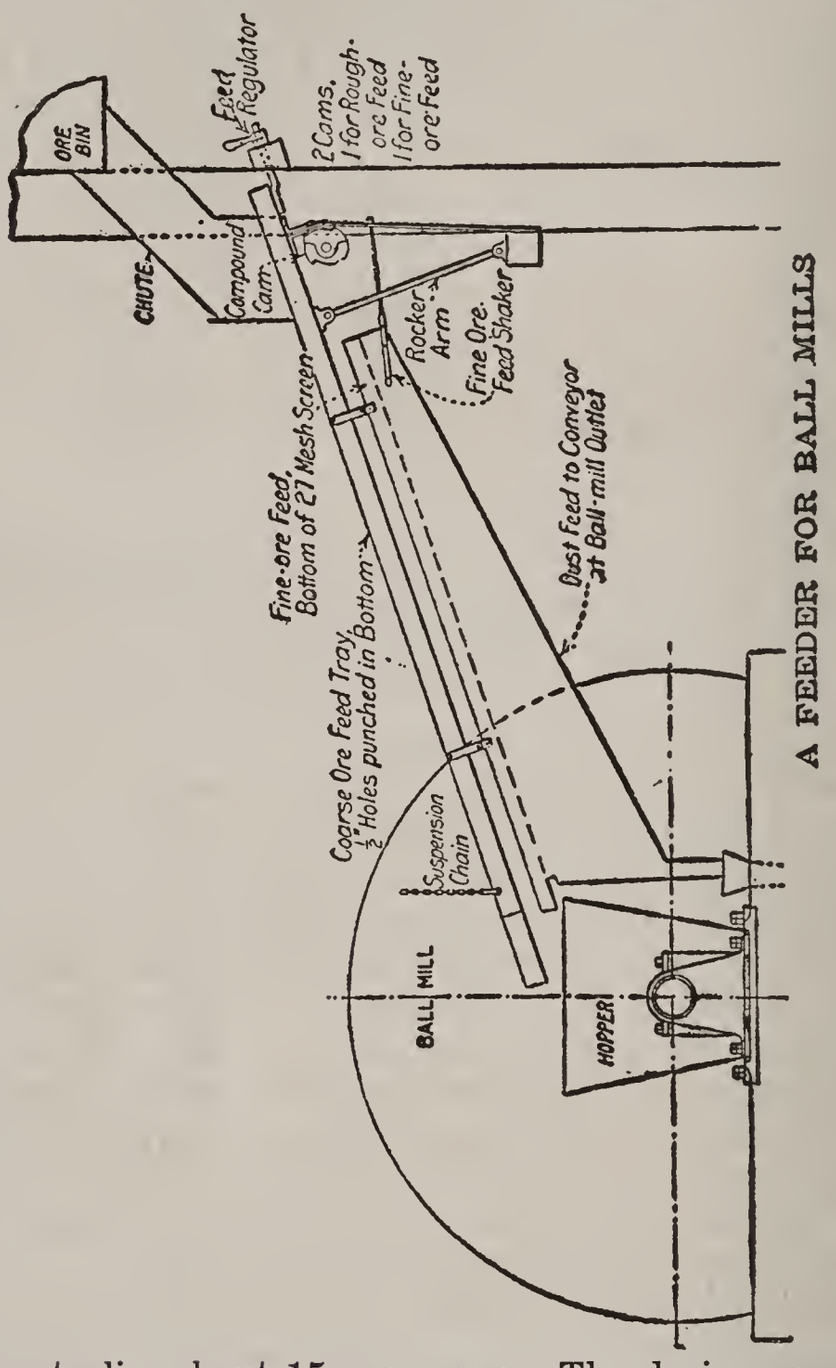
which is found to be the most satisfactory, although by measuring the concentration of acid the sulphide may be oxidized partially to sulphuric acid. Accordingly, the acid present must be carefully regulated to the stated quantity. The iodate solution should be about 0.05 molar, and may be readily standardized by weighing, and is found to keep indefinitely. In the titration a bottle devised in this laboratory may be used to good advantage, a description of which will soon be published by O. D. Neal in the *Journal* of the American Chemical Society.

It seems possible that, according to Linden and Rector, stannic sulphide might absorb H_2S and thus give high results, but our results have not shown this to be the case, and tests on other sulphides by Mr. Dean have shown that the amount absorbed is negligible.

While we have not made investigations over a sufficiently wide range to recommend the method for technical use, it seems possible that further work might find it capable of practical application. The method has the advantage of a permanent standard, and free access of air causes no difficulty since strange as it may seem, iodide solutions are perfectly stable in strong hydrochloric acid.

A Ball-Mill Feeder

The present activity in ball-milling practice suggests the publication of details of a type of feeder that was erected and successfully operated at Kalgoorlie, West-

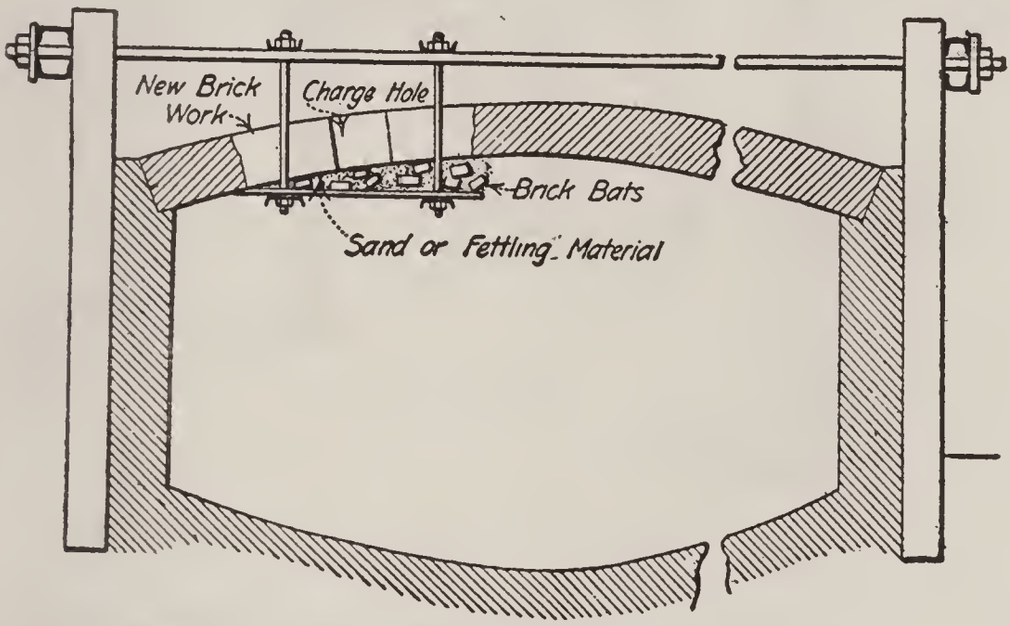


A FEEDER FOR BALL MILLS

ern Australia, about 15 years ago. The design may not be widely known, but it possesses several interesting features, and although used in connection with dry crushing in this instance, it would be equally applicable under many other conditions. At the South Kalgurli mine, where this feeder was in operation, it was found that from 10 to 15 per cent of ore was bypassed to the ball-mill outlet, thus effecting a considerable increase in milling efficiency. The upper tray consists of steel plate perforated with ½-in. holes. The fine screen below corresponds in aperture width to the one used in the mill (Krupp type), in this case 27 mesh. A compound cam gives the necessary jar to both screens and provides for a greater frequency in the vibration of the finer screen than in the punched plate. The details and drawings are taken from R. Allen's "West Australian Metallurgical Practice."

Replacing a Reverberatory-Furnace Charge Hole

A steel-and-sand support is employed at the Steptoe plant of the Nevada Consolidated Copper Co., at McGill, Nev., in replacing burned-out charge holes in the roof of a reverberatory furnace. The brick work around the charge hole is always the first to be destroyed in the course of the furnace campaign. After cutting back the burned brick far enough to obtain a suitable bearing, the use of the method shown in the accompanying



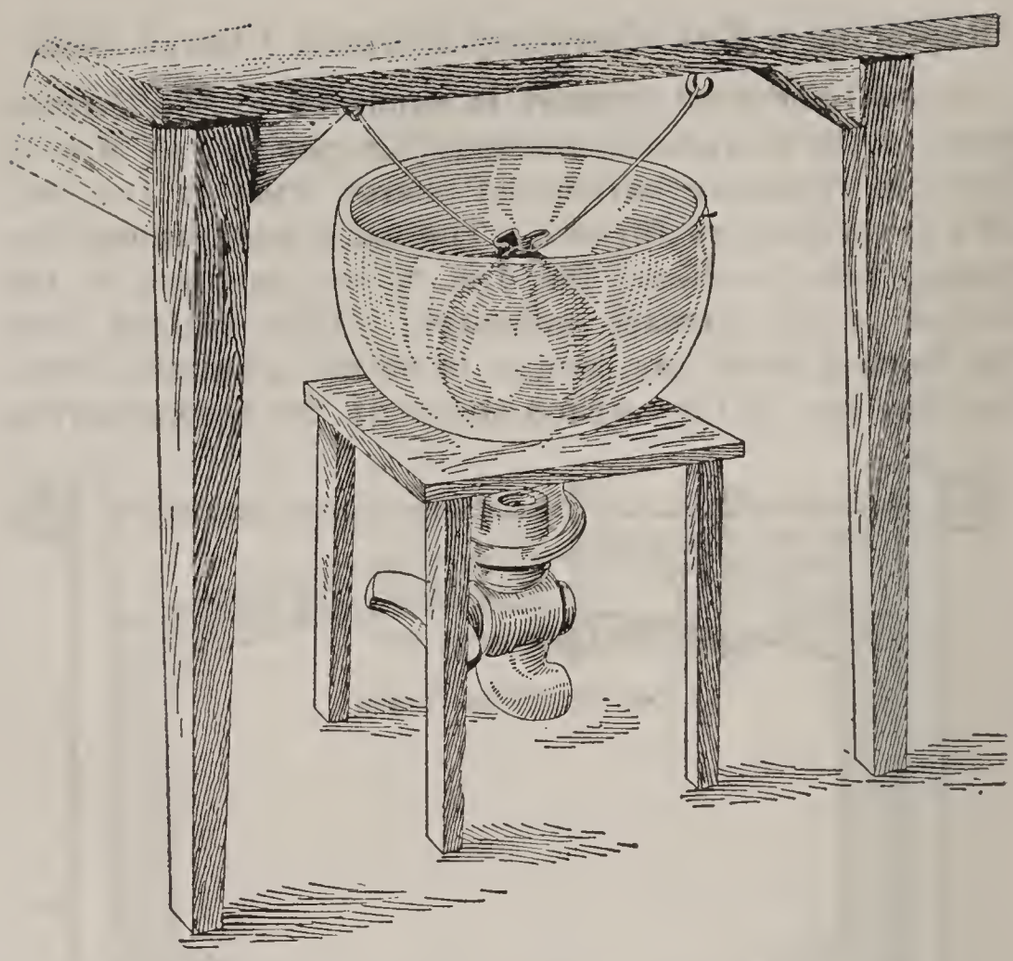
REPAIRING FURNACE ROOF AT McGILL, NEV.

sketch makes it possible to replace the brickwork around the charge hole in an interval of three or four hours, while with the older method of building a wooden "center" supported on the furnace hearth it was necessary to allow the furnace to cool off for several days. The hanging steel-and-sand "center" is dropped on the hearth of the furnace after the brickwork has been keyed up and the several parts are pulled out through the side doors before firing up the furnace again.

Container for Lead Acetate

At the Aurora Consolidated Mines Co., Aurora, Nev., trouble was experienced with the lead sponges in the assay of gold and silver cyanide solutions by the zinc-dust method. Since using the container herein described, the trouble has been overcome and good consistent sponges are obtained.

Cut the bottom out of a glass carboy and fit the mouth with a stoneware stop-cock. Invert the carboy and place it on a stand. It can then be placed under a table out of the way. A bag of sheeting or thin canvas is made to hold the lead acetate. Place about 10 lb. of lead acetate, crushed to ½ in., in the bag which should be suspended from the underside of the table

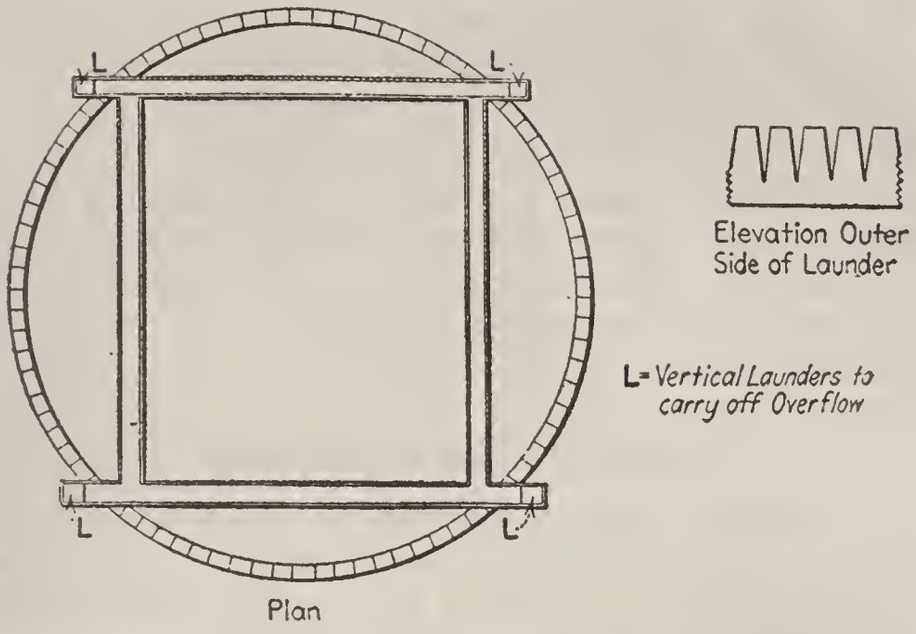


LEAD ACETATE IS KEPT IN BAG HUNG IN CARBOY

so as not to rest on the bottom of the carboy. Fill the latter half full of water. At the end of 24 hours the solution will be strong enough for use. It will need no more attention until nearly used up, when the carboy and sack should be cleaned out and fresh acetate and water put in. Thus a supply of clear and saturated solution is always ready. With 30 solution-assays per day, replenishing is necessary about once a month.

Altering Dorr Agitators

In a Western milling plant, three Dorr agitators were installed along with other cyaniding machinery. After the plant was put into operation, it was found that the gold went into solution so readily in the crushing department that no agitation was necessary, and further, that the settling capacity of the plant was decidedly insufficient. It was decided to change the three agitators into thickeners. This was done by extending the feed launders from the sides to the centers of the tanks, removing the revolving pulp-distributing arms, providing an outlet in the center of the bottom of each tank and adding decanting launders. The latter consisted of four straight launders in the shape of a rectangle on each tank, as shown in the illustration. This form of construction is much simpler than the usual annular launder at the top of the tank. A row of notches was

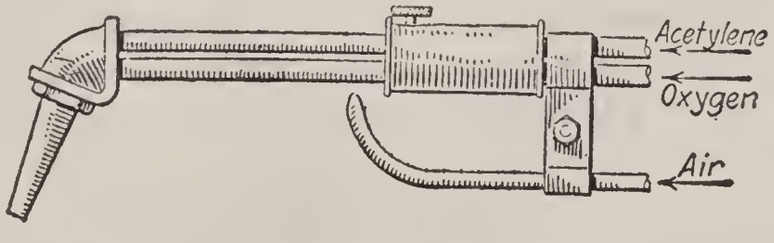


ARRANGEMENT OF LAUNDERS IN THICKENING TANK

cut along the outer side of each launder, but not to the bottom, permitting the clear liquid on top of the tank to flow through them and along the launders to the vertical discharge launders at the corners. In addition to being easier and cheaper to construct than the circular launders, the rectangular form served a direct purpose. The composition of the ore was such that an excessive amount of foam collected on top of the thickeners, the foam containing an appreciable percentage of soluble matter. The rectangular launder arrangement made a trap in which this foam was caught and from which it was skimmed at regular intervals.

Oxyacetylene Welding Kink

An oxyacetylene welding equipment, such as has been widely and successfully used heretofore, described as consisting of an air hose and a short piece of plugged pipe with a number of small holes drilled in it. The pipe is laid on the bench or work under the hands of the welder, and the air protects his hands and face from the intense heat. The disadvantages of this device are that it does not follow the movements of the worker, is easily shifted or knocked off the bench or work and, if laid on the heated article being welded, tends to cool it.

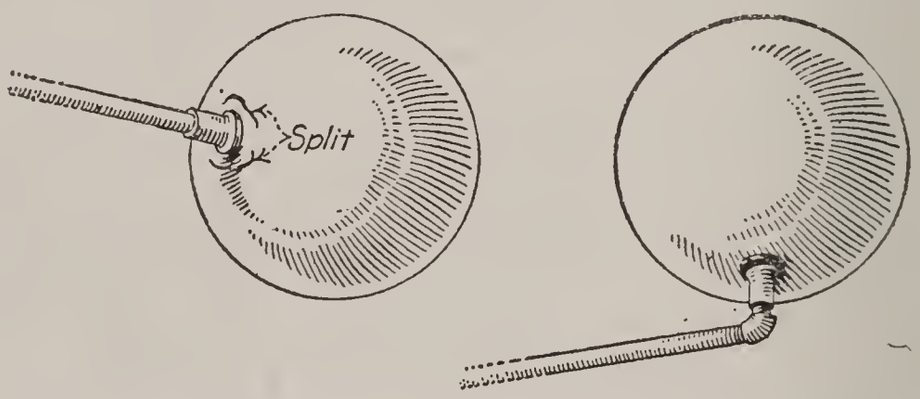


OXYACETYLENE WELDING TORCH WITH PIPE
An apparatus easily handled

The illustration here shown is an arrangement consisting of a ¼-in. pipe connected by a light hose to the air line. The pipe is bent to surround the hand of the welder and is fastened by a light bent-iron clip to the torch. By this means the welder can keep his hands and face cool without any of the disadvantages of the other method. The arrangement works well and is satisfactory from all points of view.

Improved Float Connection

In connection with a number of large tanks, float-controlled water regulators were used, and the valves being of large size, the floats were called upon to exert quite a lot of force. Considerable trouble was caused by the floats giving way around the socket to which the arm was attached, the float and rod being placed in a horizontal line. To correct this, one was turned to a vertical position so that the float pulled at right angles



FLOAT CONNECTED TO AVOID DESTRUCTIVE STRAIN

and the lifting stress was distributed around the entire circumference of the float. This was found to cure the trouble, and since changing all our floats, no trouble from splitting has been experienced.

A New Method of Expressing Protective Alkalinity

The following is what I believe to be a new method of expressing protective alkalinity in cyanide solutions. It seems to have some points that make it preferable to the old way of expressing it as so many pounds of CaO per ton of solution. This method and the accompanying chart were worked up by me last December and have been in use here since. The protective alkalinity is expressed as a percentage value.

From the equations,

$$2KCN + H_2SO_4 = 2HCN + K_2SO_4$$

and

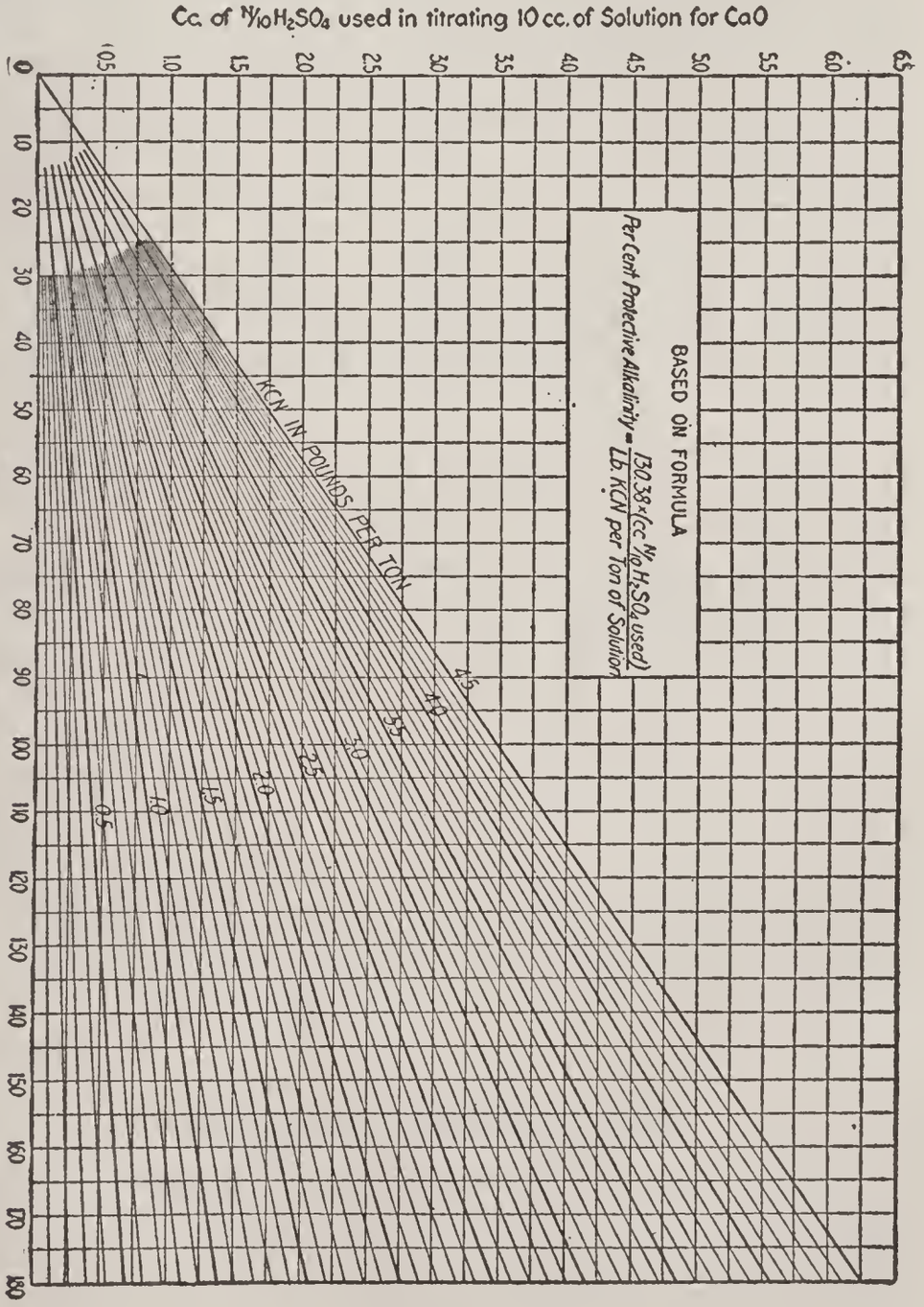
$$CaO + H_2SO_4 = CaSO_4 + H_2O$$

we get a factor for changing alkalinity due to CaO and to its equivalent alkalinity due to KCN. This factor is CaO = 2.32245KCN. This theoretical factor checks experimentally fairly closely if phenolphthalein is used as indicator; if methyl orange or other indicators are used, the carbonates, etc., in the water throw the result off a good deal.

If we use N/10 H_2SO_4 for titrating and take 10 c.c. of solution, then every c.c. of H_2SO_4 used represents 0.5614 lb. of CaO per ton of solution. Hence our formula for protective alkalinity becomes

$$\textit{Per cent protective alkalinity} = 100 \times 2.32245 \times 0.5614 \times \frac{\textit{c.c. N/10 } H_2SO_4 \textit{ used}}{\textit{lb. KCN per ton}}$$

A concrete example of how to use the accompanying chart may not be amiss. Ten c.c. of solution is titrated for KCN by the familiar $AgNO_3$ titration with KI indicator. After excess $AgNO_3$ is indicated, an excess of potassium ferrocyanide is added to prevent any zinc



THE PROTECTIVE ALKALINITY CHART

present from precipitating as zinc hydrate. Then the N/10 H_2SO_4 titration with phenolphthalein indicator is performed. Suppose that our solution titrated 2.5 lb. KCN per ton and that it required 2.0 c.c. N/10 H_2SO_4 to destroy the red color. Go up on the left edge of the chart to 2.0 c.c. H_2SO_4, then horizontally across to meet the diagonal 2.5 lb. KCN line, then from the intersection of these two lines go down vertically to read 104 per cent protective alkalinity. This method gives, in my opinion, a much clearer idea of the relative protective alkalinities in solutions of various cyanide strengths than does the usual method of expressing it as merely so many pounds of CaO per ton of solution. If, as is the case here, NaCN is used in place of KCN, the formula and chart still hold, provided the cyanide present is expressed in terms of KCN.

Use of Building Paper in Merrill Presses

The frames of the Merrill triangular precipitation presses are generally covered with canvas and a light, cheap twill—the latter being removed and burned, or dissolved in the sulphuric-acid treatment, to recover any precipitate on it. With the increasing cost of cloths during these times a lower priced and efficient substitute was sought. The practice of the Alaska-Treadwell company of putting a filter paper next to the twill was adopted. At Hedley, B. C., one sheet of unglazed building paper of good grade is used next to the twill. The cost of the paper sheets is not over 2c. a piece, while the cost of the twill is from 17.5 to 22.5c. The twills will last several months when thus protected by the building paper.

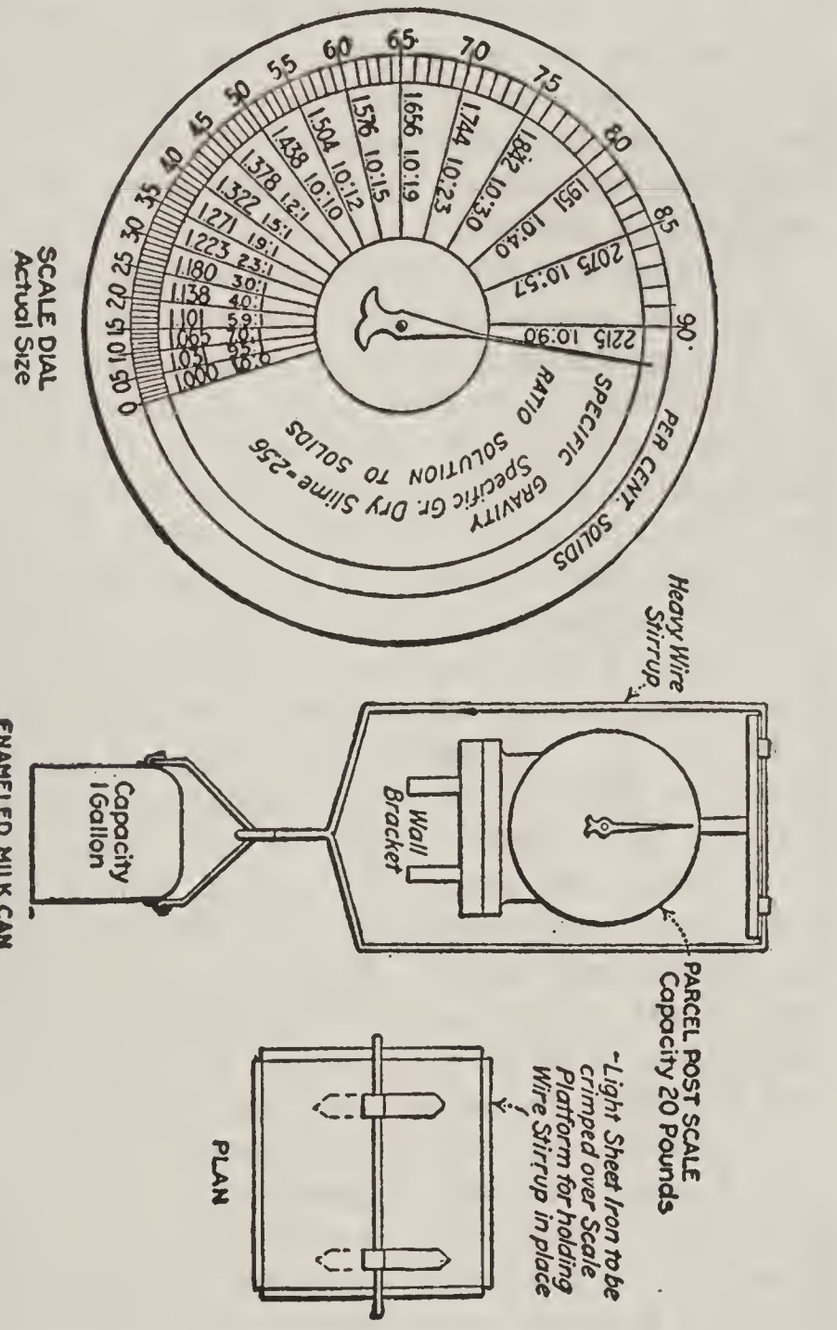
At each cleanup, the paper with all the precipitate clinging to it drops into a cloth suspended under the entire length of the press and above a tray for added protection. The twills are left in place, clean and untouched. The cloth containing the precipitate and papers is removed to the acid vat where the contents are dumped into the vat and the cloth kept for future cleanups. When it becomes too rich with precipitate it is also placed in the acid vat to be dissolved. The method is clean, quick, and inexpensive where the acid treatment is used for zinc-dust precipitates.

Specific Gravity Apparatus

One having had experience with the specific gravity apparatus in general use about cyanide plants will at once see the advantage of direct readings and the par-

ticular kind of apparatus shown in the illustration. The accuracy is well within the limits possible in sampling mill pulps. In this instance a parcel-post scale was used.

I thought it advisable for our purpose to improve on the apparatus described by J. M. Lilligren by having the readings direct in percentage of solids, and in addition to the specific gravity, the ratio of solids to solution are shown also. A piece of drafting paper may



THE SPECIFIC-GRAVITY APPARATUS

be cut to the circle of the dial, graduated and pasted thereon or, better still, a piece of tracing cloth can be cut to the circle of the dial and held there while the points for the graduations are laid off on it. The transparency of the cloth will facilitate the drafting. It can then be removed and the calibrations inked-in over the points. From this blueprints can be made for any number of scales around a plant.

Laying off the graduations for any particular ore, where factors of five in the percentage-of-solids group are desired, may be accomplished either by the aid of a

set of specific gravity tables or with the following formula:

$$\textit{Per cent. of solids} = \frac{S(a-1)100}{a(S-1)}$$

where S = sp.gr. of dry slime and a = sp.gr. of wet pulp, S to be determined beforehand in the usual way.

The percentage-of-solids figures are assumed to be 5, 10, 15, 20, . . . 90 and the equation solved for a by substituting the percentage-of-solids in each case and the specific gravity of the dry slime. Having previously decided on the size of sample to be taken in practice, the type and capacity of scale and the kind of container best suited to accurate work, the procedure would be as follows:

Assuming a 1-gal. sample and a scale of 20 lb. capacity and an enameled milk can, the starting point on the dial would be 8 lb. 5 oz. (1 gal. water weighs 8.33 lb. at 60 deg. F.). This point on the dial corresponds to sp.gr. = 1 or 0 per cent solids on the tracing cloth. Substituting 5 per cent and solving for a we get 1.031. Then 1.031 $\times$ 8.33 = 8.58 lb. = 8 lb. 9 oz., which is the second point on the new dial. Similarly 10 per cent solids is equivalent to sp.gr. 1.065, and 1.065 $\times$ 8.33 = 8.87 lb. = 8 lb. 14 oz., or the third point on the new dial. After laying off the points on the new dial from 5, 10, 15, . . . and 90 per cent and drawing radial lines through them, the intermediate points, as between 0 and 5, 5 and 10, 10 and 15, etc., may be obtained by dividing the respective spaces into 5 equal parts without seriously affecting the accuracy of those points for routine mill work. If it seems desirable that the neck be left on the can and the capacity is greater than one gallon, it is only necessary to adjust the scale to read zero with the empty can suspended from the hook. After filling the can to overflowing with water, the weight may be noted and substituted for 8.33 lb. in the calculations. After the new dial is pasted in place, the can should be filled with water while suspended from the hook and adjustments made on the thumb-nut to bring the pointer over sp.gr. 1 and 0 per cent solids. The scale is then ready for use.

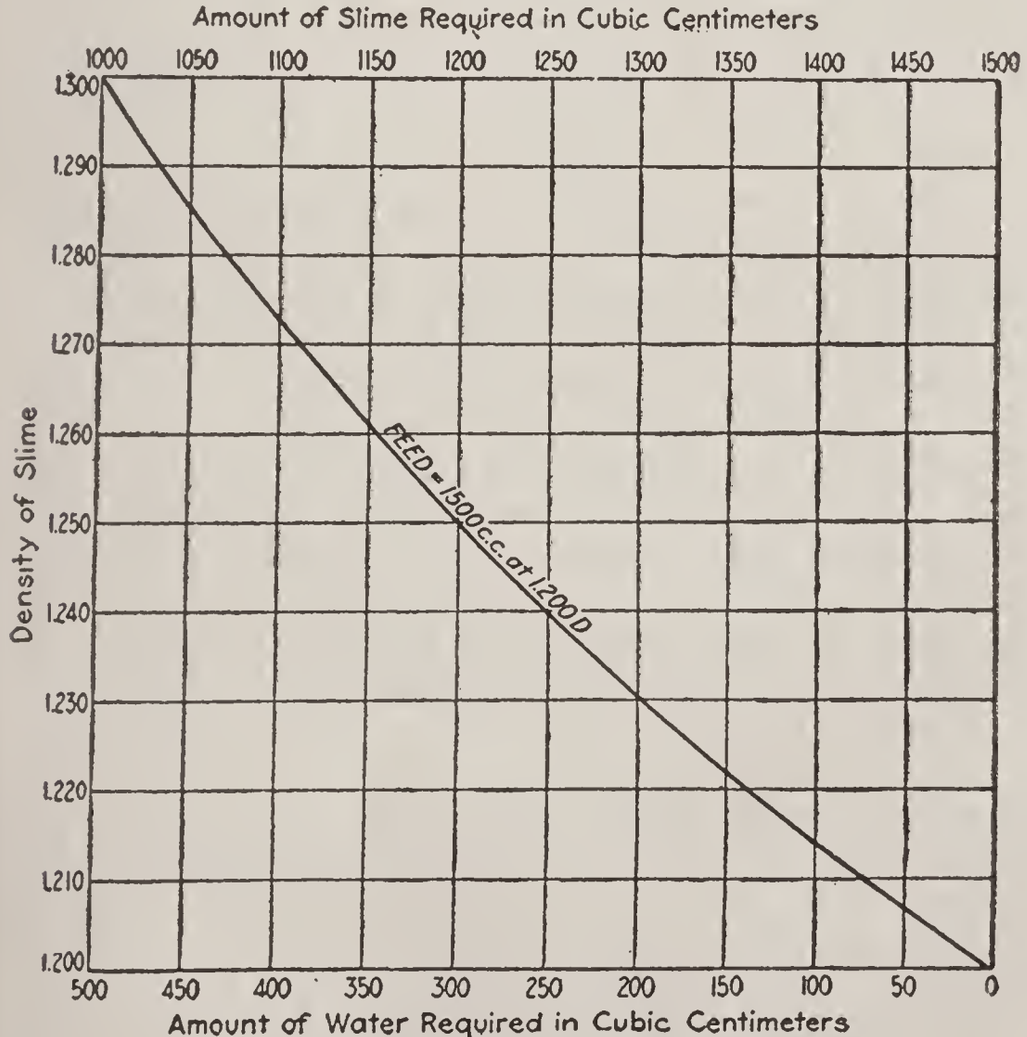
It should be kept in mind that the percentage-of-solids group of figures applies for pulps in which the dry slime is of fairly constant specific gravity. This, of course, is true for specific-gravity tables as well. Obviously, dry ore-slime from different localities and having different specific gravities would require a calibration in percentage of solids for each particular ore. Whenever a lime coating collects on the can, dilute hydrochloric acid will remove it without injury to the

can, providing the enamel has not been cracked. Or the scales can be adjusted by means of the thumb-nut so that the pointer will read 0 per cent solids or sp.gr. 1 when the can is full of water, the reduction in the cubical contents due to the lime coating being negligible when the coating is not abnormal. The cans we buy hold one gallon with the top flange or neck removed which facilitates mill determinations because the can is simply filled to overflowing.

Chart for Flotation Testing

When testing flotation oils, it is often necessary to have a feed of constant assay, volume and density. It is also sometimes essential to use a feed in the laboratory machine that is identical with the feed used in actual practice, so far as possible. Some of the difficulties I have encountered, and the remedies applied when testing oils, are herein described.

Having about one hundred oils to test out as to their frothing and selective qualities, as compared to the oil used in regular practice at the mill, I collected about 200 lb. of the feed to the flotation machines, dried it and passed all through a 150-mesh screen. This was then thoroughly mixed, quartered and assayed. To compare the results of the mill machine with the labor-



FEED CHART FOR FLOTATION TESTING

atory machine (a Janney testing machine), I used the same oil that was being used in the mill, a charge of 500 grams and a volume of 1,500 c.c. I was greatly surprised to find that the results with the Janney were much lower than we were getting in actual practice. I then tried mixing feeds of various densities and used various amounts of oil, but in every case the concentrates were low and the tails high. I next tried drying more of the slimes at low temperatures, and again I got the same results. Adding a few drops of sulphuric acid lowered the tails but also lowered the grade of concentrates. This led to the conclusion that no matter how low the temperature was in drying, the galena and pyrite took on a film of oxide that interfered with the results when testing. The sulphuric acid probably removed this film, also brought the calcium and magnesium carbonates to the top with the concentrates.

The mill slimes are mixed in Dorr tanks, and in one of the tanks I found the slimes to be free from old oil. Numerous assays and density determinations on these slimes showed the lead content to be practically constant, and the density to be from 1.250 to 1.280. Collecting some of the slimes and running them with the same oil as used before, I found my results from the Janney laboratory machine to check closely with those of the mill machine. Further experiments showed that a volume of 1,500 c.c. and a density of 1.2 gave the best feed for the Janney machine. As the density of the slimes from the Dorr tank was always above 1.2, but variable, I plotted a curve that showed the quantity of slime necessary at various densities and the amount of water needed, for a constant feed of 1,500 c.c. volume and density of 1.2. On several occasions afterward I needed to change the volume and density of the feed in making other tests, so I have derived a formula that may be applied to the original curve readings to determine the volumes of slimes and water needed for any other feed of a different volume and density. This curve and the accompanying formulas will be a great time saver to any experimenter meeting conditions similar to mine. The reduction formula may be used for any volume and density, providing the original slime density is not less than that required in the feed or greater than the curve is plotted for. Caution should be used, however, in the mixing of the final feed, as the solids settle rapidly in the slimes, and there is apt to be a loss in pouring, causing the final density to be lower than that calculated for. The use of large-mouthed bottles in mixing, rapid pouring and due care will eliminate this source of error.

The diagram and formulas will show the method

used for plotting the curve, and the same method may be used for plotting any other curve that the experimenter should choose to adopt as a standard. (See equations 1, 2 and 4.) The reduction formula (see equation 3) explains itself. Numerous experiments have found the specific gravity of dry pulp in the slime herein mentioned to be 2.84 (S in equations 1 and 2). In equations 1 and 2, m is 463 grams.

Curve Formulas

Let

m = Weight of dry material in charge;
V = Volume of charge, taken as 1,500 c.c.;
W = Weight of a unit volume of water, taken as 1;
S = Specific gravity of dry material (for slime, $S = 2.84$);
P = Density of charge, taken as 1.2.

Then

$$m = \frac{V W S (P-1)}{S-1} \qquad (1)$$

To find points of curve,

Let

d = Density of slime;
Q = Amount of slime required to make the charge;
V-Q = Amount of water required for charge;

Then

$$Q = \frac{m(S-1)}{S(d-1)} \qquad (2)$$

In this particular case $Q = \frac{300}{d-1}$

To find the amount of slime required to mix a feed of any other density and volume, from this curve, let

v = Amount of slime required, as shown on curve, of any density "d";
V_1 = Volume of new charge;
D_1 = Density of new charge;
Q_1 = Amount of slime needed for the new charge;
V_1-Q_1 = Amount of water needed for the new charge;

Then

$$Q_1 = \frac{v(D_1-1)V_1}{(D-1)V} \qquad (3)$$

In this particular case,

$$Q_1 = \frac{v(D_1-1)V_1}{300}$$

A general equation, eliminating m, for determining the amount of slime and water necessary for any charge, has been contributed by H. Rabling, and is as follows:

$$Q_1 = \frac{V_1(D_1 - 1)}{d - 1} \qquad (4)$$

Miami Reverberatory Furnaces

In constructing the reverberatory furnaces at the International Smelting Co.'s Miami, Ariz., plant, the following method of bottom construction was followed to eliminate any possibility of trouble in starting.

Broken slag was brought in railroad cars from the Old Dominion smeltery and melted in a small blast furnace obtained by setting up a number of discarded furnace jackets and a motor-driven blower on the site of the reverberatory furnaces. The molten slag thus obtained was conducted in launders to the foundations

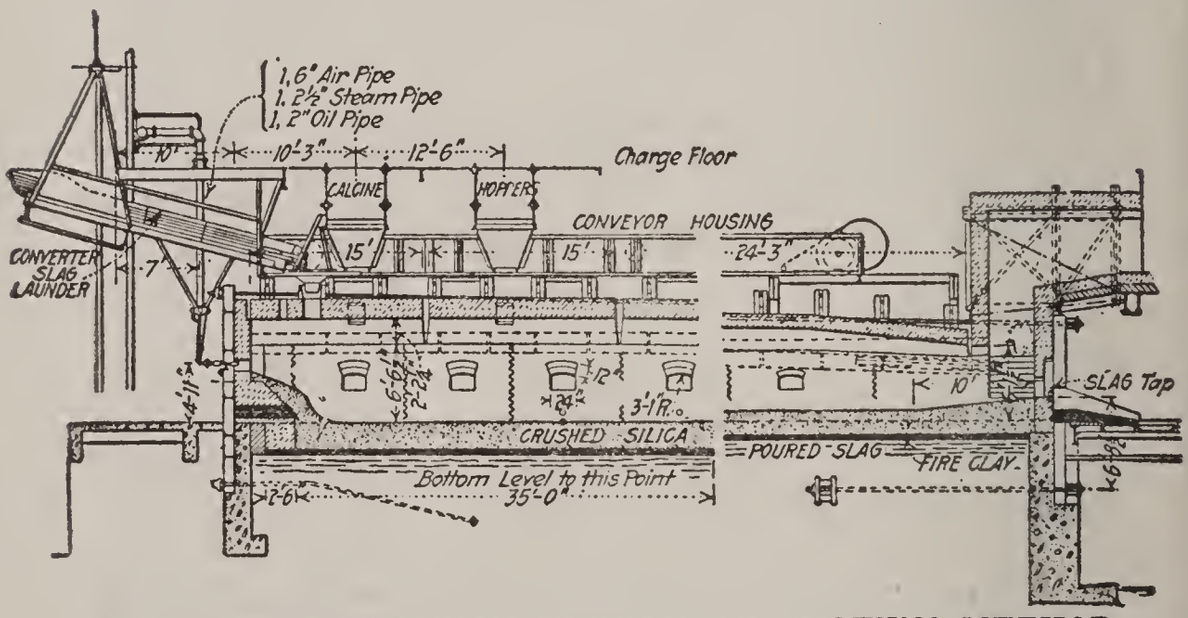


FIG. 1. SIDE ELEVATION OF FURNACE, SHOWING METHOD OF SIDE CHARGING WITH CONVEYORS

of the reverberatory furnaces. Heavy concrete beams and struts were provided between the furnaces for taking the thrust from the lower ends of the buckstays. Later, the spaces between the concrete struts and the beams were filled in with molten slag obtained from the regular operation of the reverberatory furnaces. The slag bottom was covered first with a 4-in. layer of fire-clay and then with a 27-in. layer of silica (94 per cent SiO_2) crushed to minus $\frac{1}{4}$-in. These bottoms gave no trouble whatever in starting up.

The method of side charging, instead of fettling only, along the side walls of reverberatory furnaces, now so successfully used by the Canadian Copper Co. and the Anaconda Copper Mining Co., was developed after the

construction of this plant was well along. The plant had been laid out for charge tracks running at right angles to the furnaces near the firing end. In order to distribute the charge along the sides of the furnaces, change hoppers were placed, as shown at the left of Fig. 1, under the charge tracks directly over the side

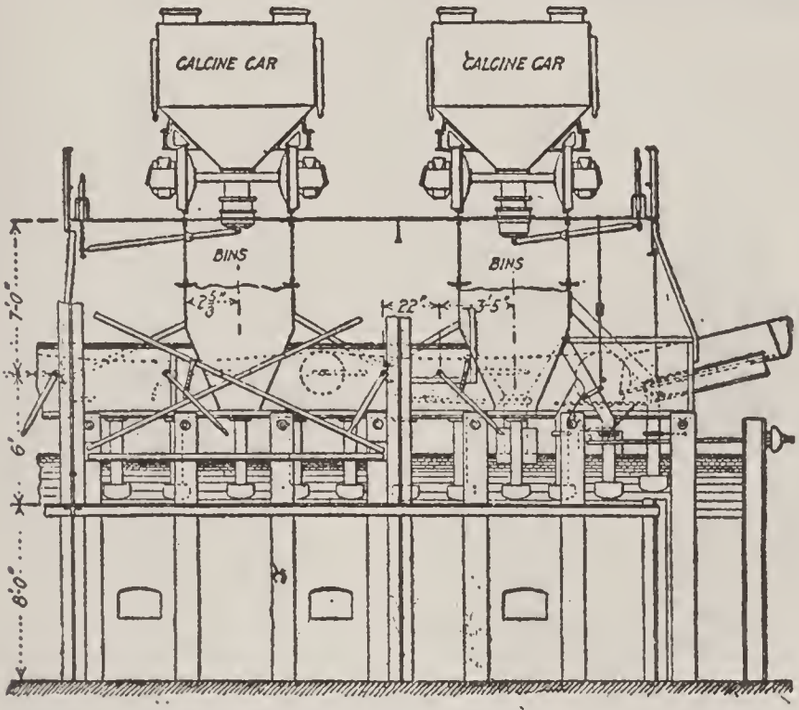


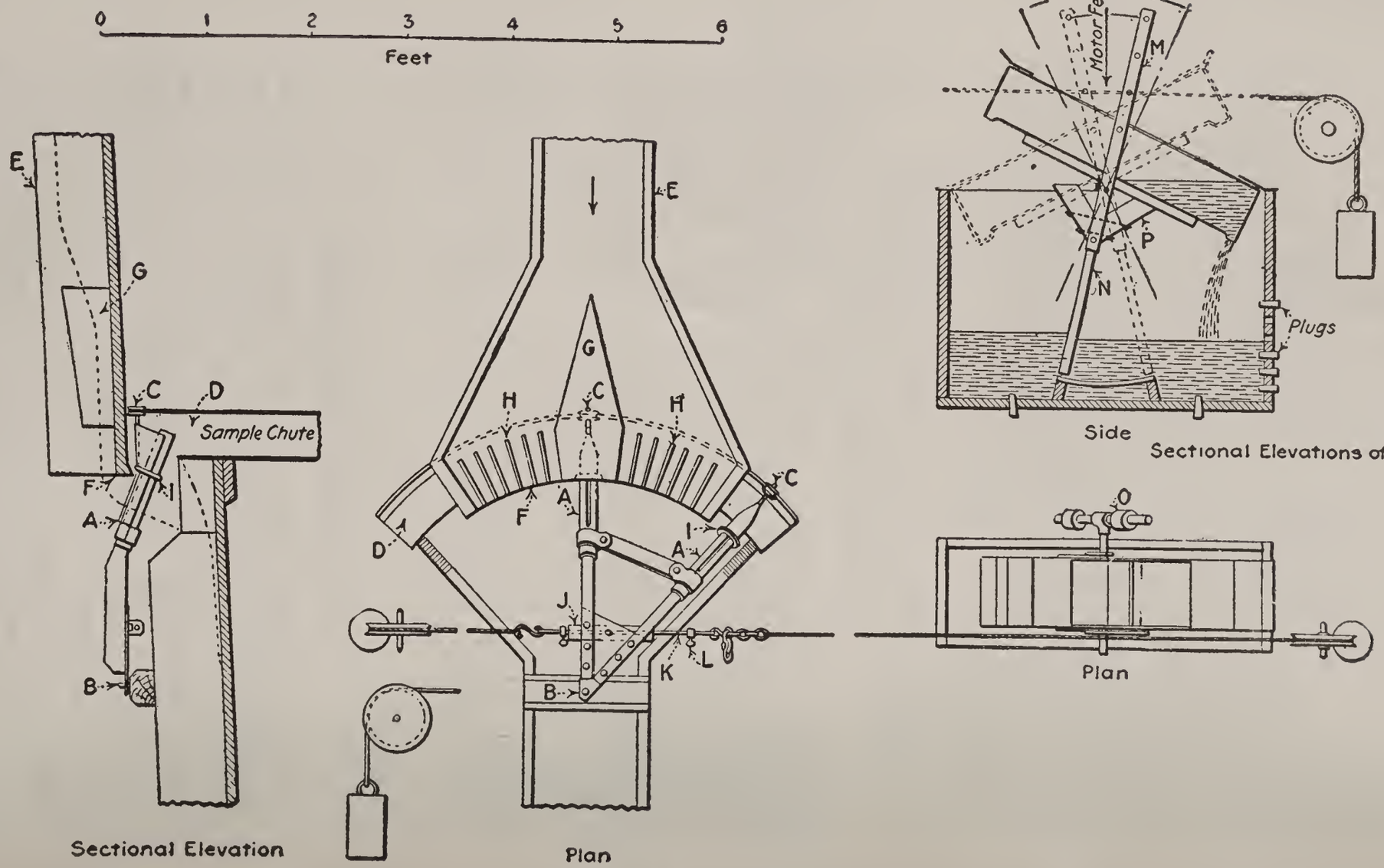
FIG. 2. LONGITUDINAL SECTION, SHOWING CHARGING ARRANGEMENT

walls of the furnaces. Drag-chain conveyors were installed, one over each side of each furnace, which received the charge from the charge hoppers. Under these conveyors, approximately every 30 in., suitable down-spouts with gates were provided, as shown in Fig. 2, so that the charge may be distributed along the side walls of the furnaces throughout their length. The bridge-wall and side walls at the firing end of the furnaces are charged by drawing directly from the hoppers through suitable spouts. The floor over the charge tracks around the skimming end of the furnace was paved with firebrick.

An Automatic Pulp Sampler

A pulp sampler that can be depended upon for accurate work and yet which may be readily constructed by a carpenter and blacksmith is shown in the accompanying illustration. The instrument is of the traveling-slot type, the motor a modification of the well-known tilting box, and the drive a wire stretched between two pendent weights.

The two traveling slots *A* are carried on the triangular framework pivoted at *B*, whose opposite extremities are supported above a circular track by the rollers *C*. The slots are of equal width, cut in heavy galvan-



FRANCIS AUTOMATIC PULP SAMPLER

ized sheet-iron tubes of 2-in. diameter, with the edges of the slots turned up to provide cutting edges.

As best shown in the sectional elevation of the sampler, the slotted tubes fit over wooden arms attached to the center pivot plate of $\frac{1}{16}$-in. iron. The supporting rollers *C* revolve on stout wire nails used for spindles, which are fastened to the lower ends of the slotted tubes by the metal saddle pieces soldered to the tubes, as shown. The open ends of the tubes discharge into the semi-circular sample chute *D*, as the triangular frame, actuated by the drive wire, revolves about its pivot, causing the slots to oscillate beneath the pulp stream flowing from the launder *E*.

It is contended that half the trick of accurate sampling consists in the proper disposition of the pulp stream previous to the sampler cut. If the stream is too deep, its surface velocity, particularly at the launder discharge lip, becomes excessive and is the cause of the worst loss by splash. A depth of 2 in. is suggested as a good general limit. In the sampler illustrated the depth of the pulp stream is reduced by widening the supply launder, as shown, as it approaches the discharge lip *F*, splitting the current with the wedge-shaped obstacle *G* and furthering the more equal distribution of the stream along the periphery of the discharge lip by the vertical radial guides *H*.

Important details that should be noted in the sectional elevation of the sampler are the receding section of the under edge of the discharge lip *F* and the circular-plate drip ring *I*, which prevent contamination of the sample by drip from the exterior of the slotted tube.

Movement of the sampler tubes beneath the pulp stream is effected by the drive wire, which is attached to the ends of the socket bar *J*, of $\frac{1}{4}$ x $1\frac{1}{4}$-in. iron, which revolves freely upon its pivot pin without play. The socket-bar ends are turned down at right angles and pierced with a $\frac{1}{2}$-in. hole to allow the free passage of the $\frac{3}{8}$-in. drawbar *K*, whose movements in the socket bar are regulated by the setscrews *L*. These setscrews are used after the motor has started up, to effect the final adjustments for stroke so that the sampler frame and tilting box bump home simultaneously, regardless of any previous movement of the drive wire, thus accomplishing satisfactory sampling movement.

The most noteworthy feature of the tilting-box motor, as best shown in the end view of the illustration, is that the power lever *M*, which is attached to the drive wire, is screwed to an independent paddle board *N*, which swings freely, within limits, about the axle of the tilting box. The axle of the tilting box is rigidly

attached to it and carries on one side a vertically set balance rod *O*, which is of ¼-in. pipe and provided with sliding weights adjustable by setscrews. With these weights the sensitiveness of the tilting box, as a whole, can be controlled to a nicety, and consequently the water charge per stroke and the strength of wire pull are easily adjusted to the work.

The tilting box illustrated is 7 x 8 x 34 in. and capable of a maximum pull of about 20 lb. Only the last half of the full downward stroke of the box is utilized for power. As the box fills with water, it falls with considerable acceleration, thus gaining power until it strikes the bracket *P*, attached to the paddle board. The force of the impact, however, is largely absorbed in overcoming the inertia of the drive wire, stretcher weights, etc. The drive-wire pull starts with the revolution of the paddle board in the water bath below. The depth of its immersion controls the speed with which the slotted tubes cut across the pulp stream, and this depth of immersion is regulated by the plugs seen in the side sectional elevation of the illustration. About an inch of clearance is allowed between the paddle board and casing as a bypass for the water to regain its level ere the return stroke.

The machine illustrated was built to sample a stream 5 in. deep by 12 in. wide. This particular design effects a stroke of about 5 in. at the drawbar, the travel of the drive wire being 1 to 2 in. in excess of this.

In sampling the pulp stream it is imperative that the cut should not be made too rapidly—there must be no bat and ball effect. It is probable that 1 yd. per sec. is the highest safe speed to insure accuracy of cut—less speed would be safer.

Pouring Assay Melts Upon a Flat Plate

The conventional conical iron molds have been universally used by assayers for the reception of assay melts, and while the method of pouring upon a flat iron plate is by no means new, strangely enough a description of it does not appear in the literature on assaying.

Upon the completion of fusion the lead button rests on the bottom of the crucible, with the molten slag above. During the time the crucible remains in the furnace after this stage is reached, the most favorable conditions exist for the complete separation of the lead from the slag by settling. When the fusion is poured into a conical mold, the slag largely leaves the crucible first, then the lead, and following the lead the small

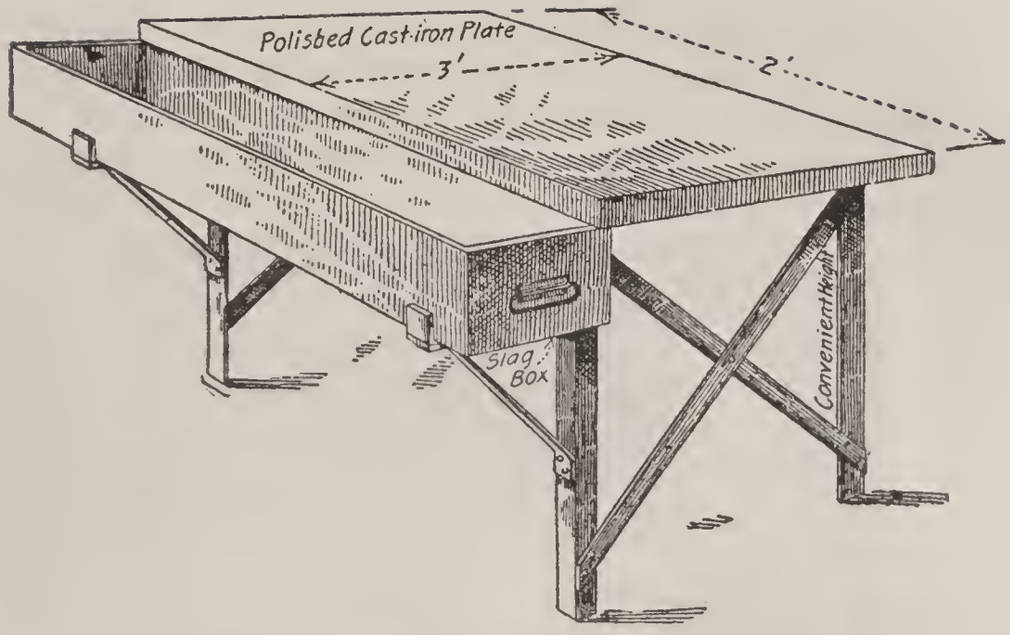


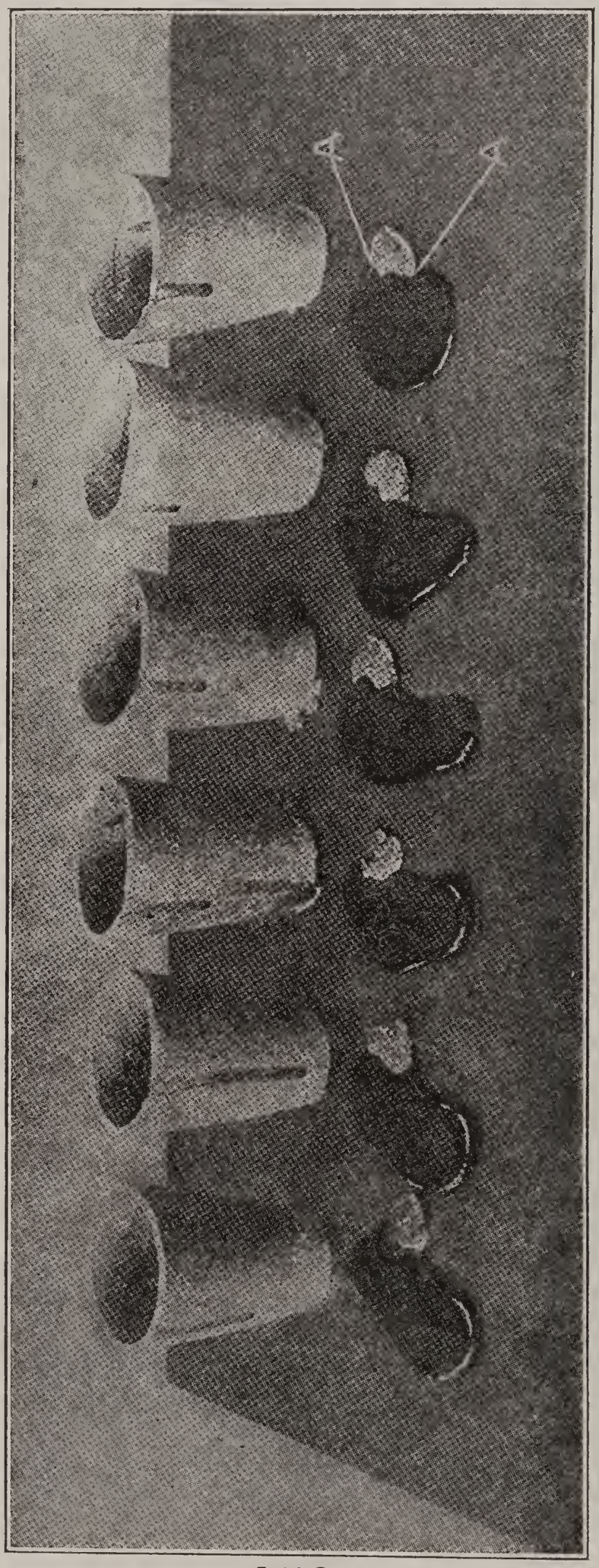
FIG. 1. STAND AND FLAT PLATE

amount of remaining slag. At the moment the molten slag strikes the conical mold there is necessarily a slight chilling effect, which causes the thin layer of slag in contact with the iron mold to become viscous, if not actually solid. Into the slightly chilled large mass of slag falls the lead button, which at once sinks to the bottom of the mold, where it finds a resting place on the thin layer of highly chilled slag. As the molten lead falls through the slag, opportunity is afforded for the entrainment of small particles of lead. However, with properly proportioned charges, if sufficiently hot when poured, there is little tendency for the slag to retain shot lead. When the solidified melt is removed from the mold, the sides of the button are found to be incased in a thin layer of slag, and the top of the button is more or less firmly attached to the large mass of slag. The union between slag and button at this point is sometimes so strong that in removing the slag a thin layer of the button is taken with it. In any event a considerable amount of hammering is necessary to free such a button from the adhering slag.

Use of a Flat, Polished Iron Plate Found Advisable

It is evident that if a method of pouring were used in which the relative position of the lead and the slag was not changed during pouring, there would be less likelihood of particles of the lead being entrained in the slag, and moreover, a button would be obtained which would be much more free from slag. I have found that pouring upon a flat iron plate largely accomplishes this end, as well as presenting certain other advantages. After experimenting with plates set at different angles, as well as with special plates having depressions of various shapes for the accommodation of the button, it was found that a flat cast-iron plate, which of course must

be smooth and preferable polished, gives the best results. The plate can be mounted upon a gas-pipe or angle-iron rack of convenient height, as shown in Fig. 1. At one end is a pair of brackets which support a sheet-iron box into which the rejected slag is swept by



a counter brush. Before using for the first lot of fusions, the plate should be slightly heated. In the absence of other means, this may be readily accomplished by pouring a sufficient amount of gasoline on it to more or less cover the surface, igniting and wiping off with a rag the thin film of water that remains after the gasoline burns. After the first fusions have been poured, no further attention need be paid to the temperature of the plate, as the succeeding pourings keep it sufficiently warm. When pouring, the crucible should be held in such a position that the slag may be observed as it flows from the crucible. After the major portion of the slag has left the crucible and the lead becomes visible, the crucible should be drawn slightly toward the operator, so that when the lead runs out, it does not fall directly into the mass of the slag, but strikes the edge of the partly solidified slag next to the operator, which acts as a dam, breaking the force of its fall. A little practice with this apparently difficult manipulation makes one entirely proficient in its operation. The appearance of a set of melts poured upon a flat plate is shown in Fig. 2. A plate of moderate size, say 2 x 3 ft. will accommodate 24 melts.

In a surprisingly short time both the lead and the slag are solidified. The button may therefore be picked up for cupellation almost at once, whereas with the conical mold a considerable time is necessary for the lead and the slag to solidify, and then not infrequently, the assayer in his haste will disturb the melt before the lead is solid. Even after the slag is solid and apparently the whole mass in the mold is solid, the lead may still be liquid, because of its lower melting point. The buttons resulting from pouring on a flat plate when the operation is properly conducted are in one mass, and in general are perfectly clean and free from slag with the exception of the little rim or margin of the button *AA* in Fig. 2, which is directly in contact with the slag. All that is necessary to prepare the buttons for cupellation is to bend them over so that they can be placed in the cupel. Time spent in forming the perfect cubes with carefully flattened corners by hammering, which is a tradition of assaying, is merely wasted, as there is no slag to remove—hence little hammering is required. Another difficulty that is sometimes encountered with the conical mold, that is not apparent with the flat plate is the sticking of the slag and button to the molds. When the slag once begins to stick to a mold, this difficulty gradually becomes worse until the mold is thoroughly cleaned and scoured. If, through accident, the slag should contain beads of lead, these are far more apparent in the thin cake of slag when held up to the

light than in the more compact mass of slag resulting from the use of the conical mold, and can be separated and recovered easily.

A somewhat greater degree of skill is necessary in pouring upon a flat plate, and furthermore, the pouring requires a slightly longer time than when using the conical mold, but the other advantages more than compensate for these slight drawbacks.

Colorimetric Determination of Bismuth

No originality is claimed for the following method of estimating bismuth in refined copper, but it is submitted as a contribution that may be of interest to chemists. I am prompted to give it by the fact that after many years in different countries I have found it in use in only two places. Over 12 years ago it was employed at Wallaroo, South Australia, where possibly the process was worked out. A method employing the same principle but differing in detail was in use at the laboratory of the Tharsis company, in Glasgow, a number of years ago. Beringer, in his "Text Book of Assaying," gives a colorimetric method for bismuth, in which the solutions to be matched for shade are of a clear brown color. Health, in his "Analyses of Copper," refers to this method, but makes no mention of one using lead iodide. A method employing lead salts in the final solution has never been described in print to my knowledge. Matching light shades of yellow-brown in a clear solution is an operation of extreme delicacy; and it is a much easier matter to see a tint of brown against the pure yellow of lead iodide.

For this determination the following solutions are of convenient strength: Potassium iodide, 17 grams per liter of water; lead nitrate, 13.5 grams per liter of water; bismuth standard, 0.2324 gram bismuth nitrate, $Bi(NO_3)_3 5H_2O$, per liter. One cubic centimeter of the latter equals 0.0001 gram bismuth, so that if 10 grams of copper are taken for assay, each cubic centimeter required will equal 0.001 per cent bismuth. The bismuth must be obtained in a solution free from copper, arsenic and iron and with a known amount of lead present. As the method is used for small quantities only, it is well to have a clear idea of the amount of bismuth present in the copper, so as to be able to weigh up a suitable quantity.

With this done, dissolve it in HNO_3, dilute and cool. Add Na_2CO_3 to precipitate a small quantity of copper, in the cold, and stir occasionally for six hours. Filter off the precipitate of copper, which will carry all the

bismuth. Dissolve in hydrochloric acid and precipitate with H_2S. Filter and wash well to free completely from iron. Dissolve the sulphide precipitate in HNO_3 and add 5 c.c. of the lead-nitrate solution. Precipitate by adding ammonia until solution is almost neutralized; then add an excess of ammonium carbonate. Boil, cool somewhat, add more carbonate and decant through filter. Dissolve in MNO_3 and repeat the precipitation. If the color due to copper is strong, add weak cyanide solution slowly until the blue has almost faded. Filter and wash free of copper. Wash the precipitate from the paper into a beaker and dissolve in the least quantity of weak HNO_3 by running the acid over the paper on the funnel and draining into the beaker with the precipitate.

Carefully evaporate to a syrupy state so that, on removing from the hot plate, the contents will crystallize. Cool, add three drops of HNO_3 and 5 c.c. of water, warm sufficiently to dissolve all crystals. Cool and pour into a Nessler tube up to the 25-c.c. mark. Add potassium iodide solution up to the 50-c.c. mark. Compare the shade with that obtained with the standard bismuth solution in the presence of 5 c.c. of the lead solution and an amount of HNO_3 similar to that in the beaker after evaporation. As the color darkens on standing, it is necessary to have several tubes of standards of different strengths prepared so that all shall be treated the same length of time. A good plan is to have the Nessler tubes in line and measure into each 5 c.c. of lead-nitrate solution and three drops of nitric acid; then into four of them put four different quantities of the bismuth standard solution.

If cathode copper carrying about 0.001 per cent of bismuth is being analyzed and a 10-gram sample has been taken, measure 0.6 c.c. of bismuth standard solution into No. 1 tube, 0.8 c.c. into No. 2, 1 c.c. into No. 3 and 1.2 c.c. into No. 4. Make all up to the 25-c.c. mark and then with the potassium iodide solution fill to the 50-c.c. mark at the same time that the tube containing the test assay is being filled. The color nearest to that of the test assay may be quickly picked out, the two tubes given a swirling motion and the color judged by looking down into them. As lead salts with potassium iodide give a pure yellow precipitate and bismuth salts alone give a brown coloration, it is essential to have the same amount of lead present in each tube; otherwise, for the same amount of bismuth, the one with most lead will appear lighter in shade. Warmer temperatures give lighter colors, apparently indicating lower results. Presence of HCl and NH_4Cl destroys the color.

If the HNO_3 solution is baked dry and then taken up

with water only, the bismuth will not go into solution and on addition of potassium iodide a pure yellow will be obtained. If much antimony is present, it is rendered insoluble on evaporation to dryness with the HNO_3 thereby causing retention of some bismuth giving low results. If much arsenic, copper, or iron is present, it will tend to liberate iodine, causing high results, but if the solution is cool and no more acid used than recommended, small amounts will not interfere noticeably. An excess of HNO_3 or other oxidizing agent present will liberate iodine. It has been suggested to add sodium sulphite. This must be done carefully, as too much will produce a white precipitate and destroy the color of lead iodide, giving a lighter shade by dilution of brown with white and by deficiency of yellow.

The method is useful for detecting bismuth deposited with the copper in electrolytic determinations. The following has been found satisfactory: Treat the sample of cathode copper with HNO_3 and boil the solution. Dilute to about 300 c.c. Add 10 c.c. of the lead-nitrate solution and precipitate out the carbonate as described. Dissolve in HNO_3 and proceed as before.

Assaying Gold in Copper Matte

The method described is the result of a number of years' experience in assaying copper matte running from 10 to 50 oz. Au, 20 to 200 oz. Ag and 30 to 50 per cent Cu. During this period we averaged several carload lots per week and the sample of every lot was submitted to a certain well-known umpire. The method in all essential details has been in use for three years, and we have had no difficulty in winning our share of the umpires.

At first we attempted to assay this material by the ordinary crucible method with an excess litharge charge. This is universal practice in copper smelteries for the determination of gold and silver in matte. It calls for a 20-gram crucible (which has a capacity of about 175 c.c.) charged as follows: Sodium carbonate, 12 grams; litharge, 80 grams; silica, 6 grams; niter, 5 grams; matte to be assayed, $\frac{1}{4}$ a. t. This is stirred and the following cover is added: Litharge, 40 grams; borax glass, 15 grams; sodium carbonate, 12 grams. The muffle should be hot enough to insure a quiet fusion in 20 min. This method is quite satisfactory for low-grade mattes, but for the rich material in question the slag losses and cupel absorption were too high and the method was found entirely too erratic and unreliable.

Several scorification methods were tried. One called for several (five or ten) portions of $\frac{1}{20}$ a. t. each scorified in $2\frac{1}{2}$-in. Bartlett scorifiers, the buttons to be

cupelled separately and the beads combined in one parting cup. We abandoned this method for several reasons, the principal one being that it offers excellent opportunities for the multiplication of weighing errors at the pulp balance.

In another standard method, ¼ a. t. portions are scorified in 3-in. Bartlett scorifiers with about 85 grams of test lead. The matte is mixed with one-half the lead, after which the rest of the lead and a little borax glass are added for a cover. The assays are run in pairs, and the buttons from the first scorification are combined, made up to 65 grams with test lead, fluxed with 1 gram of silica and rescorified. The buttons still contain too much copper for cupellation and must be rescorified a second time. The results obtained by this method are certainly above criticism, but unfortunately, results are not always obtained. The temperature required for the first scorification is over 1,100 deg. C., and I have been unable to find furnaces, muffles or scorifiers that would stand this heat for continued use. A slight drop in temperature at the start causes some constituent of the charge to separate out and float, effectively covering the lead and spoiling the assay. The muffles rapidly collapse, and the loss of scorifiers is extremely high. Nothing can be more discouraging than to nurse along an assay under such conditions only to have it leak out through a crack in a scorifier in the third scorification. It entails a serious loss of time and material, the former particularly annoying.

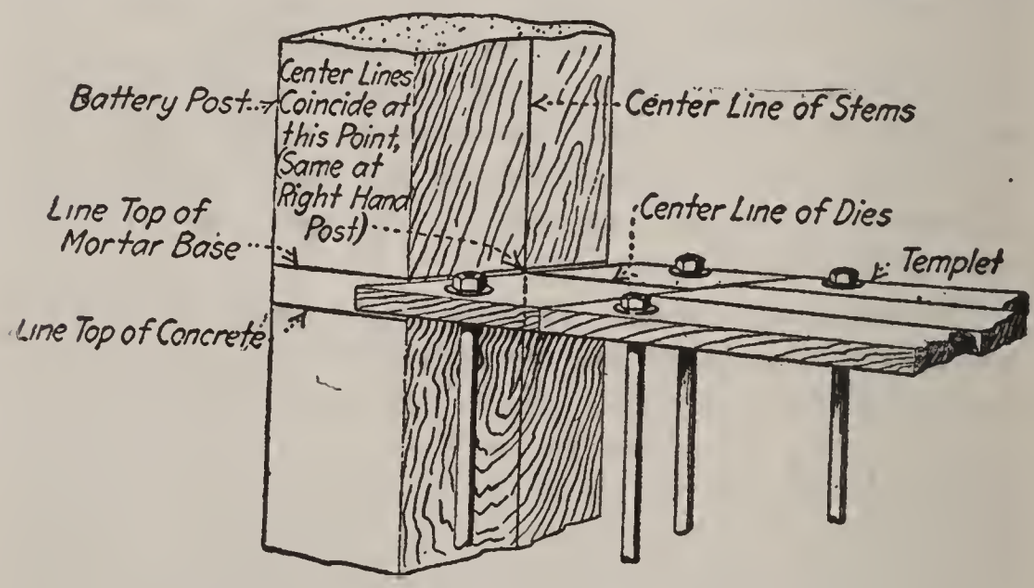
To overcome these various difficulties we finally adopted a scheme of our own, which is carried out in the following manner: Place 35 grams of test lead and 2 grams of silica in each of four 3-in. Bartlett scorifiers. Weigh out ¼ a.t. of the matte to be assayed and, by tapping the edge of the scale pan with the spatula, drop approximately one-half into each of two scorifiers. Repeat this operation with the other two scorifiers. (At first we made ⅛ a. t. weights by filling the bottoms of some 5-gram weights, but we found the method of splitting quarters to be faster and more accurate.) Stir and add 35 grams test lead and 2 grams borax glass as a cover. Open and scorify hot. Combine the buttons two and two and make up to 65 grams with test lead. Add 2 grams of silica and rescorify as before. The buttons will weigh about 20 grams and will cupel with feathers.

Flatten the beads on an anvil, combine them two and two in porcelain annealing cups and heat for 20 min. in dilute (1-5) nitric acid. Pour off the acid and heat 10 or 15 min. with nitric acid of 1.30 sp.gr. We use electric stoves wound so as to give a heat just sufficient to boil

acid in these cups. Wash the beads first in hot dilute ammonia and then in distilled water. Dry and anneal. Each cup now contains, ready for weighing, gold from ½ a. t. This method has been found to be accurate, rapid, economical and easily worked.

Replacing Mortar Blocks

The mortar blocks in stamp mills were formerly made of planks bolted or spiked together; but nowadays concrete has replaced lumber for many reasons, among which the high first cost and short life are the chief factors. It is safe to assume the average life of wooden mortar blocks to be eight years. This may be considered to be nearly the life of the first stage of the mine, but many mines last much longer. Wood will age and rot whether the mill is in operation or not. In most of the mortar blocks replaced the trouble was found to have started from the bottom. On account of the sidehill location of the mills the pits were practically a drain for the surface water and mill water, which soon caused the wood to rot. Wood preservatives, when used, prolong the life of the wood, but their use is not general. In



LINING UP STAMPS WITH A TEMPLET

some blocks where the planks were put lengthwise with the mortar, the outside rotten planks may be changed, but it is noticed that the block needs repairing within a very short time. Taking all factors into consideration, concrete mortar blocks are the most serviceable.

The first thing to do toward placing concrete mortar blocks is to get the mortar out of the way. The stamps must be raised as far as possible and held firmly. This may be accomplished by loosening the tappets and sliding them on the stem, or by putting a block on the finger jack. In many cases they will not be raised far enough, and the mortar will have to be moved away by taking the stamps out and putting them on the camshaft floor. The mortar is raised off the bolts with

wedges and crowbars helped with chain blocks tied to the camshaft or battery girts. As this work always takes place after a clean-up, it is advisable to make the mortar as light as possible and to keep the dies and liners out.

To take the mortar away a strong cribbing is made of timbers not less than 4 x 6 in., with two heavy pieces (about 6 x 8 in. or 8 x 8 in.) laid lengthwise on the top, horizontally. On the top of the two heavy timbers lay bars of ½ x 3-in. flat iron, extending all the way under the mortar to keep the rollers from sinking into the wood. Do not forget that a five-stamp mortar weighs about three tons. With the mortar raised entirely clear of the bolts, place rollers made of 1-in. or 1½-in. round iron the entire length of the mortar. Then roll the mortar block away on the cribbing with the help of rope or chain blocks, helping and guiding the crowbars. Raise the mortar block as far as possible until it touches the girt, and hold it there with strong dependable chains tied to the girt and camshaft. This avoids the trouble of moving the mortar away and gives ample room for working in most cases. The next step is to take out the old wooden blocks. At first this seems easy, but any millman who has done it once will think differently. The planks extending above the floor are handled easily enough, but after that part has been disposed of the wood for a depth of 6 to 8 ft. has to be removed. Recalling several instances, I remember blocks set in solid concrete which had to be drilled and blasted just as in sinking a shaft. Augers were tried, but they did not work well in wet wood. The best results obtained were with holes 12 to 16 in. deep loaded with one-third of a stick of ⅞-in. 40 per cent dynamite.

In another case the block was set with concrete on one side, solid rock on the other. After we had blasted 2 ft. of the block (this block rotted from the top) the wood was so fibrous that more holes for blasting were impossible; so we sunk a small shaft next to the block in the rock, which was easier to drill than the concrete, and got at it from the side. As quickly as the bottom is reached in any small place, the rest is quickly taken out and the most difficult part of the work has been accomplished. In an old 10-stamp mill the blocks looked very good from the outside; but when the mill was started up after years of idleness, the blocks sank at the rate of an inch a day. When taken out they were found to be sound for the first 6 ft., but the remaining 6 ft. was dug out with a spade.

The next step consists of putting in concrete blocks. The place is well cleaned of all wood and loose rock. According to the size of the rocks to be put in and the

nature of the ground, it would seem advisable to bore holes in the rock on the bottom and sides and cement in rods or pipes for the purpose of anchoring the block to the rock.

Two lines are drawn horizontally on the battery post —one to mark the top of the concrete, the other the top of the mortar base (sole plate). A templet is made of 2-in. boards, the holes traced and if possible bored from the mortar. The center line of the stems is extended to the bottom of each battery post, while on the templet is traced the center line of the dies. To be sure that the shoes will strike exactly on the dies, the center line of the die stems produced should be coincident with the center line of the shoes as marked on the templet. The templet holding the mortar bolts in place is leveled and held firmly to the battery posts, the bottom of the nuts on the same elevation with the line of the mortar-base top.

The concrete is now ready to be mixed. A large pile of rock is wheeled near by. The mixing platform has been erected as near as possible, so that the mixed concrete can be shoveled from the platform directly into the hole. The mixture should be 1:3:5 for the lower part with coarse rock; as the height is increased, the mixture is made richer; 1:2:3 in practice has proved very good for the upper part. The concrete must be well rammed until the water comes to the surface, and reinforced with iron rods or old steel cable. I have used tailings of 20 to 40 mesh, free from slime, for sand with very good results; and crushed stone from the waste dump or sorted from the crusher that has passed a 1-in. screen has proved satisfactory for the rock. The mud-sills and battery posts can be concreted in with the mortar blocks. The same anchor bolts are used for the mortar, anchored into the concrete with an iron bar or a large rail, one for every two bolts with large square cast-iron washers. The top of the mortar block is finished with a mixture of one part cement and two of sand, $\frac{1}{2}$ in. thick.

This finishing must be done the day after the concrete has been put in, to remove the chances of the finish scaling off after the mill is put in operation, due to an imperfect bond. To straighten and level the top before it gets too hard, use a piece of square steel about 5 ft. long (a bar of heavy pick steel is very handy, as it is straight and the edge is sharp) with a level on top. The bar is used as a scraper to shave off the high spots.

Ample time is given the concrete to set. It requires two weeks for a block of such size to harden. It would be disastrous to try to save a few days and have the block crack. In dry weather it is advisable to cover the

concrete with gunny sacks and to keep them wet for four or five days, thus preventing a too rapid setting of the outside portion. The forms are taken off 10 days after the concrete is put in, as there is less danger of breaking a corner when the concrete is still green.

If not damaged, the same rubber pad used on the wooden block is used for the concrete. The purpose of it is not to act as a cushion, but as packing to cause a true fit of the mortar. The mortar is set in about the same manner as it is taken out, and repair work is done to the battery if needed.

The cost of exchanging the wooden for concrete mortar blocks in a 10-stamp mill was as follows:

Labor:		Hr.
Raising mortar and tearing out old floor		16
Taking off wooden blocks and drilling holes for anchor bolts		60
Screening dump for rock, sand, tailings wheeled to mill		72
Mixing and pouring concrete		96
Replacing mortar on new blocks and readjustment of batteries		24
		268
268 hr. at 47c		$125.96
Material:		
Reinforcing rods (from scrap pile)	.....	
Lumber for forms (borrowed and returned to mine pile)	.....	
60 sacks cement @ 80c	$48.00	
Freight, 3 tons @ $20	60.00	
		$108.00
Total cost		$233.96

While you have everything handy for mixing concrete and while your back and muscles are trained for this sort of work, put in a concrete floor for the tables, if you have not one already, as it is a great improvement over the warped and leaky planks usually found in floors. Six to 8 in. of concrete on a well-tamped ground with 1 in. of fine to finish will make a good floor. It can be inclined to drain all to one place. This floor should not touch the mortar block or battery post. About a 2-in. space should be left and filled with sand to prevent the transmission of vibrations to the tables, which would cause the floor to break in many places.

Treating Amalgamation Tailings

The tailings resulting from old pan-amalgamation mills have been difficult to treat economically—in fact, it is extremely doubtful whether any profit has been made by a well-known company that has treated many thousands of tons of tailings from the Comstock Lode, Nevada.

The tailings contain a great deal of organic matter, including charcoal. The silver contained is in the form of chloride and sulphide. The mercury exists as a mercurous salt for the most part, although there is some metallic. By careful panning about 1 per cent of clean concentrate is obtained, assaying 52 oz. silver and 1.2

oz. gold per ton. The copper contained is insoluble in water, but 80 per cent can be dissolved by sodium thiosulphate.

Half the mercury existing as a mercurous salt is soluble in sodium cyanide and sodium sulphocyanide with a precipitation of gray mercury.

$$Hg_2Cl_2 + 2\ KCN = Hg(CN)_2 + 2\ KCl + Hg$$

$$Hg_2Cl_2 + 2\ NaCNS = Hg(CNS)_2 + 2\ KCl + Hg$$

The mercury sulphocyanide forms a double salt with the excess of sodium sulphocyanide present. The mercury is precipitated by zinc and iron. In practice silver and gold were precipitated on pump runners and lines, pipe lines and in tube mills. The cement backing to the liners in the tube mills, ground, assayed 6.2 oz. Au and 54 oz. Ag per ton. The coarse cement scale assayed 5.2 oz. Au and 20.4 oz. Ag per ton.

Coarse material caught on screens at tube-mill discharges, containing a small amount of concentrate and a grayish precipitate, but principally consisting of coarse sand and pieces of flint pebbles, assayed 1.7 oz. Au and 9.4 oz. Ag per ton. The tailings being treated at this time assayed 0.12 oz. Au and 3.6 oz. Ag per ton.

All iron in contact with pregnant solution showed evidence of precipitation of gold and silver, the gold being more easily precipitated than the silver. The precipitation of silver and gold is caused by the mercury being precipitated on the iron and in turn acting as a precipitant for the gold and silver. Gray mercury also precipitates gold and silver from cyanide solutions, but in practice I do not think the small amount present interferes very much with the extraction of the gold and silver, as any precipitated would be dissolved again. However, with weak cyanide solutions, relatively poor extractions are made in the same time as are made with strong solutions, say 0.04 per cent NaCN and 0.2 per cent NaCN. With a strong solution, treating unground tailings, 48 hr. agitation, a washed residue assaying 0.01 oz. Au and 0.9 oz. Ag is easily obtained; the cyanide consumption is about 0.5 lb. per ton more than when using a weaker solution. It may be that any prematurely precipitated gold and silver would not be dissolved by weak solutions in an ordinary time of treatment.

As the mill solutions contain sulphocyanides, some double sulphocyanide of mercury is formed. This may explain why the addition of mercuric chloride to working solutions, as an aid to extraction in the treatment of silver ores, is not successful. In my experimental work in Mexico, using freshly prepared solutions, the addition of small quantities of mercuric chloride to the solution at the last 24 hr. of treatment always resulted

in a better extraction of the silver, while in practice after the mill had run two months, no beneficial result was observed, and the use of mercuric chloride was discontinued.

As has been stated, the mercury in the tailings is partly soluble in sodium thiosulphate, an average of a number of experiments showing that 55 per cent of the mercury could be dissolved by a 0.1 per cent solution of sodium thiosulphate. The mercury can be precipitated from the solution by means of an alkaline sulphide.

As is well known, when sodium thiosulphate is added slowly to mercury salts, the following reactions take place:

(1) $Hg_2(NO_3)_2 + Na_2S_2O_3 = Hg_2S + Na_2SO_4 + N_2O_5$

(2) $3\ Hg(NO_3)_2 + 2\ Na_2S_2O_3 = 2\ HgS.Hg(NO_3)_2 + 2\ Na_2SO_4 + 2\ N_2O_5$

(3) $3\ HgCl_2 + 2\ Na_2S_2O_3 + H_2O = HgS.HgCl_2 + 2\ Na_2SO_4 + 4\ HCl$

But when an excess of sodium thiosulphate is added, then the following reactions occur:

(4) $HgCl_2 + Na_2S_2O_3 = HgS_2O_3 + NaCl$

(5) $Hg(NO_3)_2 + Na_2S_2O_3 = HgS_2O_3 + 2\ NaNO_3$

(6) $Hg_2(NO_3)_2 + Na_2S_2O_3 = HgS_2O_3 + 2\ NaNO_3 + Hg$

(7) $Hg_2Cl_2 + Na_2S_2O_3 = HgS_2O_3 + 2\ NaCl + Hg$

The mercury thiosulphate in each case dissolves in the excess of sodium thiosulphate, forming a double salt.

The time of agitation in the experimental work varied from 2 to 12 hr., 6 hr. being found to be sufficient. The consumption of "hypo" varied from 3 to 4 lb. of commercial salt per ton of tailings treated. In a 1-hr. agitation, 29 per cent of the gold and 23.9 per cent of the silver were dissolved. Certain experiments show that it is possible to obtain extractions of the gold and silver equally as good as those obtained by cyanidation.

There are yet some 350,000 tons of untreated tailings of the class described awaiting some economical treatment. A great deal of experimental flotation work has been done by many on these tailings, and although 85 per cent of the combined value of the gold and silver has been concentrated, the cleaning of the first concentrate appears to be difficult. From some experiments made personally, to obtain a good result it was necessary to make a first concentrate from 12 to 16 per cent by weight of original charge. Before flotation the charge was given a preliminary treatment with sodium sulphide. In cleaning the concentrate, I found that ground charcoal (about 10 lb. per ton of concentrate) agitated thoroughly with the concentrate effectively assisted in the cleaning.

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